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THE

SCHOOL OF MINES QUARTERLY.

VOL. XXXII. NOVEMBER, 1910. No. 1.

FOUNDATIONS FOR THE MUNICIPAL BUILDING, NEW YORK.

BY MAURICE DEUTSCH, C.E.*

Perhaps the most interesting and difficult piece of foundation work carried on anywhere in this country is that just completed for the great Municipal Office Building for New York City. After about twelve years of planning, and in accordance with State legislation enacted in 1907, the Commissioner of Bridges was finally authorized to provide for the erection of a building on land already acquired for the extension of the Brooklyn Bridge. After competitive designs from twelve well-known architects were submitted, those of McKim, Mead & White were finally accepted. Their design provided for twenty-five main stories surmounted by a tower of fifteen additional stories. The total height of the building including the tower will be 580 feet, making it the third highest in the City. It will have a frontage of 381 feet on Centre Street, 168 feet rear on Park Row and 158½ feet deep. The total office area, exclusive of corridors and two shafts, will be more than 600,000 square feet.

The high tower is directly over Chambers Street, one of the main traffic arteries of the City, which will pass directly through the center of the building. The deepest part of the structure will be about thirty-five feet below curb. At that level will be the great terminal for the present subway and all other future underground

* With the Foundation Co.

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railroads. The building will provide offices for the principal departments of the City, and perhaps some of the courts, which are to-day paying a total rental of $453,000 annually. Its total cost will be in the neighborhood of $11,000,000.

The building will be of steel skeleton construction, fire-proof throughout, with floors and walls supported entirely by the steel

FIG. 1. Centre St. Elevation.

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superstructure, which alone weighs 25,000 tons. The total weight of the structure itself, increased by the weight of its contents and by the stresses due to wind pressure, is estimated at about 165,000 tons, all of which will be carried by 129 main columns resting on 106 pneumatic caissons sunk through water, quicksand and coarse sand and gravel to the greatest depth to which the process has yet been used for building foundations, viz., 112 feet below water level.

In the construction of the tall office buildings on Manhattan Island many serious foundation questions have arisen from time to time. Some of these have given very little difficulty, due to the fact that little excavation was required to reach bed-rock or some other hard bottom. But in the southern part of the city the soil for a considerable depth below the curb is of a very treacherous nature. Great depths of excavation were required in certain localities before striking hard strata, and these could only be reached by applying the pneumatic caisson system of foundation construction. This variety of foundation problems arises from the characteristic geology of Manhattan Island. At the northern end Manhattan schist and Fordham gneiss crop up above the street level. At such points the dip of the rock in a southerly direction is quite evident, but at the southern end of the island this same rock is found only at depths between fifty and one hundred and fifty feet below curb. Above this rock are found strata of a scientifically interesting nature. The ten-foot layer of hard-pan overlying the rock is only an accumulation of disintegrated rock and glacial drift hardened by the pressure of the superimposed material, which in excavations for the foundations of the Singer Building and United States Express Building has been found to contain specimens of shells belonging to an extinct species. The discovery of these shells has aroused great interest. By reason of the lower half of Manhattan Island being very narrow and surrounded on all sides by tidal waters, we find the soil above the hard-pan to be saturated with water. Layers of quicksand to a thickness of as much as sixty feet have been encountered in down-town excavations for foundations. On the site of the Municipal Building it was encountered only at the south end of the lot, where the caissons were carried to bed-rock. The upper layer of material in the down-town section is usually filled-in ground. Excavating through this fill to depths

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of from ten to thirty feet below street level the ground water level is encountered, which is practically the elevation of the water in New York Bay. Below this point have been found the various strata of great geological interest.

The rock surface of Manhattan Island slopes from north to south, and passes below the tide level at about Tenth Street. From this point south and below the water level the island consists of a great blanket of sand and silt deposits built up on the underlying rock by the tide and currents of recent geological times. Beds of clay and sand vary in thickness, and in some places the clay does not appear at all. The sands vary to a considerable degree from the coarsest gravel to the most treacherous quicksands. A rational analysis of quicksands such as have been excavated in lower Manhattan shows about forty per cent. of quartz, about twenty per cent. of clay and mica, and about forty per cent. of feldspathic detritus, indicating that a considerable portion of it had been the result of disintegration of granite and gneiss, the latter forming the backbone of Manhattan Island. The color also varies in different localities. On the corner of Maiden Lane and William Street (the Royal Queen Building), as well as in certain localities along the west shore, the color of the quicksand is a dark gray, while on the site of the Singer Building the color is red to reddish brown. These quicksands are a sedimentary deposit of disintegrated rock carried down from the Jersey shores and laid down by the action of the tides. The red-colored sands point to the red sandstones and the gray to the shales, which are characteristic rocks of eastern Jersey.

To give an idea of the fineness of this sand some samples were passed through sieves with meshes of various sizes, the finest passing through a 200-mesh sieve, the openings of which are about .035 of an inch in width. From 20 to 37 per cent. passed through the 200-mesh sieve. Little may we wonder then that such fine material, when saturated with water, should flow as readily as water itself, passing through every crevice and seeking its level just like any liquid. Herein lies the reason for the difficulties encountered in excavating in quicksands; for, should pumping be resorted to in order to drain a pit of water, this fine material will invariably be pumped with the water, and if the pumping be continued will finally

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be drawn from under adjoining structures, eventually undermining them.

The hard bottom upon which these sands rest consists of a cemented mass of material usually called "hard-pan," which was evidently deposited during the glacial period. Some specimens of hard-pan will often defy the eye of the expert by its resemblance to concrete.

To make clear how it has been possible to expose these deep layers of glacial drift frequently found at depths of thirty to one hundred and thirty-five feet below water level, it may be well to describe briefly the method used in sinking the caissons for the Municipal and other high buildings. The caissons are simply air-tight bottomless boxes, rectangular or cylindrical in cross-section, having interior chambers large enough for a gang of men to enter and excavate at the bottom. In the roof of the box is an opening from which a shaft communicates with the ground level, of a size sufficient for the conveyance of men and material. Compressed air is pumped into the caisson as it sinks below the water line, to expel the water from the open, lower or cutting edge. The caisson is thus sunk by undermining, aided by the heavy weight of concrete which is added over the roof of the caisson as the latter gradually sinks, and also by the addition of pig-iron blocks. When it has reached a satisfactory hard stratum, which is cleaned and levelled, the whole interior of the caisson and of the shaft connecting the working-chamber with the outer air is filled with well-rammed concrete, forming a solid monolith, upon which the superstructure is readily supported. To permit the passing of men and material from the outside air to the working-chamber, or vice versa, without excessive loss of compressed air, a Moran lock or air-chamber, invented by Mr. Daniel E. Moran, Vice-President of The Foundation Co. (Class '84 C.E.), is used. This air-lock surmounts the top of the shaft leading to the working-chamber, and it is now possible to sink caissons through quicksand and water close enough to adjoining buildings without causing a flow of material from under the latter, which if allowed to occur might settle and crack them.

Preliminary explorations were made for the city's architects by driving four wash borings and one dry boring to a depth of about 135 ft. below street level or about 103 ft. below mean high water,

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at which depth in boring No. 1, rock or boulder was encountered, and in the other three borings the maximum depth was 100 ft. below M.H.W. These preliminary borings indicated a good coarse sand to a depth of about 50 ft. below street level, becoming finer to a depth of about 85 ft. and then somewhat coarser to a depth of about 130 ft. The sand showed traces of brown loam and clay and was very similar to that found at the World and St. Paul buildings.

Based upon these borings and subsequent loading tests made on the sand at various levels up to a depth of 77 ft. below curb level, the building was designed to have 116 caissons carried down to an elevation —28 below M.H.W., or about 60 ft. below curb level, and about 22 ft. below ground water level. Caissons were to be sunk by the pneumatic process and were proportioned to impose a maximum load of 8 tons/sq. ft. on the fine sand at this depth. The contract was awarded after competitive bidding, but before work was done the Commissioner of Bridges cancelled it, owing to objections that arose from the Building Department, engineers and others throughout the city. It was finally decided to carry the 116 caissons to bed-rock, and bids were again advertised for. Seven bids were received and the contract was finally awarded to The Foundation Company of New York City for $1,443,147 for 116 caissons on bed-rock and proportioned to carry a load of 15 tons/sq. ft. In order to check these preliminary borings and to make sure of the location of bed-rock the contractors started a series of diamond drill borings which were later extended by the Department of Bridges. A total of 21 borings were made between the dates of July 22, 1909, and May 24, 1910. The results of these borings showed that the wash borings taken at the north end of the lot did not extend to rock as had been supposed, but terminated on boulders about 50 to 80 ft. above it, and that the rock sloped very abruptly toward Duane Street. The deepest bore hole on the lot occupied by the building was that of boring no. 3, at the extreme northwest corner of the building, where rock was found at an elevation of about 177.8 ft. below M.H.W., or 209 ft. below street level. The greatest depth found was that of boring no. 20, which is outside of the space occupied by the building and about 170 ft. north of it. The depth at that point is 256.9 ft. below

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M.H.W., or 288.9 ft. below street level. South of this abrupt slope covering about two thirds of the area of the building lot, the rock was approximately level and at an average depth of 130 ft. below the curb at the intersection of Chamber and Centre Streets. As the limit of the application of the pneumatic process is reached at about 100 ft. below water level it is evident that it was impossible to carry all caissons to bed-rock by this method. Other methods, such as sinking concrete cylinders by open dredging, excavating in frozen ground or by compacting the ground by forcing grout into the surrounding soil, were all carefully considered. It was found that these other methods would greatly increase the cost, and besides would otherwise be impracticable. It was finally decided to modify the caisson plan and adopt the pneumatic process as required by the original contract.

All of the piers for the south wing of the building and for the heaviest or tower portions are located where rock is nearest to the surface, so that these remained unchanged. These comprise 68 caissons, which cover about two thirds of the lot. The remaining 48 piers for the north wing of the building were located where the recent borings showed the underlying rock to be the deepest. It was decided to replace these by 38 caissons of such larger cross-section as to impose a pressure of only 6 tons/sq. ft. on the coarse sand found at a depth of 40 ft. below mean high water, or 72 ft. below street level; as compared to 8 tons/sq. ft. as originally proportioned. It is well known that when confined so that there is no tendency for flow, sand will bear very heavy loads. Furthermore, the depth decided on is far below any likely disturbance from future subterranean construction work.

Before the contractors could proceed to carry out this change of plan, it was necessary that the contract between the City and The Foundation Company be modified, as from the wording of the contract,

"The specifications are intended to cover all material and labor required to complete the caisson foundations to the levels indicated in the drawings. The caisson foundations are to be of concrete as shown on the caisson plan and to be put in place by the pneumatic process. All caissons must be carried down to rock."

When this contract was signed the parties assumed that the wash borings truly indicated the depth of the rock, and the contract

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drawings showed the caissons carried to rock at the approximate depth indicated by the preliminary borings. Furthermore, the contractors did not include any margin for experimenting with new methods for carrying foundations to depths heretofore considered impracticable. In view of these considerations, the authorities finally decided upon the composite rock-sand foundation, and the contract was then modified to provide for this adjustment. The original contract, calling for 116 caissons carried to bed-rock, provided for a lump sum for the foundations to a depth of —96 ft. and a price per cubic yard for work below that level. The total bid amounted to $1,443,147, which was comprised of the following items: (*a*) $175,370 for work not included under pneumatics; (*b*) $1,117,777 under pneumatic caisson work; (*c*) $150,000 for extra pneumatic caisson work below an elevation —96, estimated as amounting to 10,000 cubic yards.

The original contract with The Foundation Company for 116 caissons sunk to rock provided for future modifications and changes such as

"may be previously agreed to in writing between the parties hereto in a manner not materially affecting the substance thereof or increasing the price to be paid in order to carry out and complete more fully the work herein agreed to be done and performed."

In accordance with this provision the contractor agreed to sink the 38 enlarged caissons resting on sand to a depth of 40 ft. below M.H.W., or 77 ft. below street level, in accordance with the modified plans, for the same price that he would have received for carrying the 48 original caissons to rock 96 ft. below M.H.W., and so that the total cost of the work would not be increased by the modification of the plan.

The plan in Fig. 2 shows the final layout of caissons as now completed. The caissons carrying the tower columns all rest on bed-rock, and the caissons to the right of the irregular line, passing through the left half of Fig. 1, rest on bed-rock, while those to the left are on a good coarse sand at an elevation of 77 ft. below street level. This change in the foundation plan was approved by the Board of Estimate and Apportionment on December 21, 1909. The contract was revised on February 24, 1910. Owing to long discussions that arose before this change was finally approved, the work

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at the north end of the lot was held up for several months, so that the city agreed to extend the contract time ninety days.

The two objections that were raised against this sand founda-

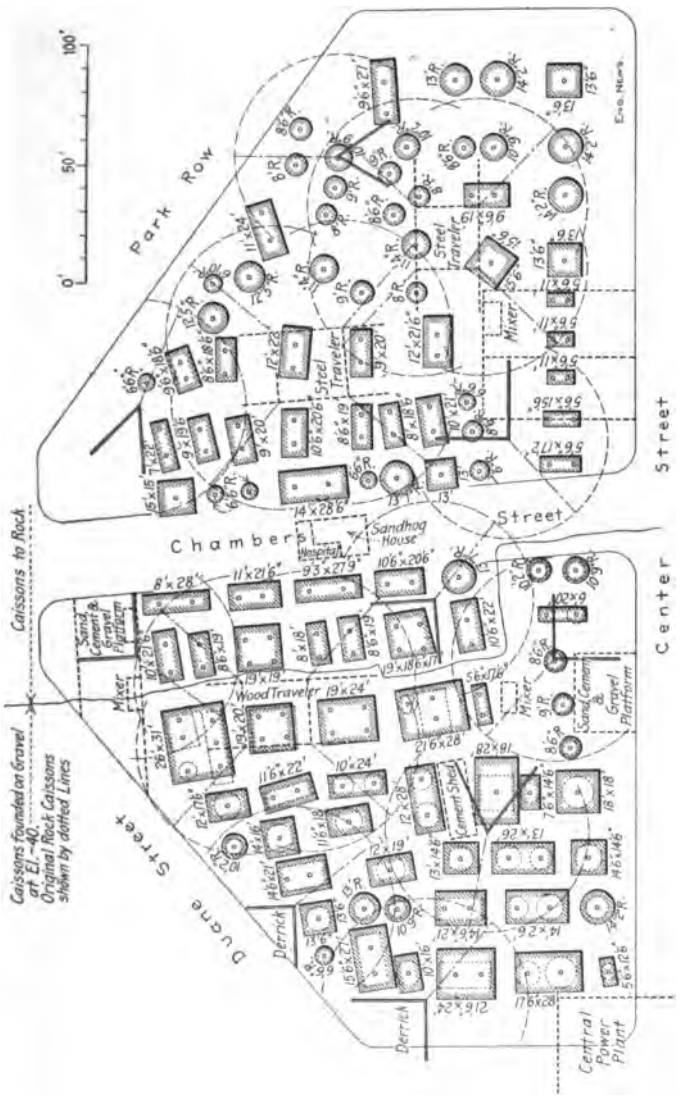

FIG. 2. Caisson plan.

tion, viz., that of a possible settlement and of a possible future disturbance, were regarded as negligible, as it was considered that a sand as much as 35 ft. below water would be secure against set-

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tlement provided the pressures per square foot on same were kept moderate. Furthermore, the depth of 77 ft. below street level was considered well below any probable future excavation in this vicinity, so that there could be no disturbance from such source.

Some idea of the precarious character of this work may be gained from the consideration that the Municipal Building site is bounded on all sides by very important and heavy structures, all of which rest on shallow foundations, while the excavation for the new building is through wet sand at depths of from 50 to 100 feet below the footings of these buildings. For example, the World Building, 15 stories in height, is only 70 ft. to the southeast, the St. Paul Building, built about twelve years ago, a 26-story steel frame structure, is only about 1,000 ft. to the southeast, and rests on a medium coarse sand at about water level, or 23 ft. below curb, the Barclay Building, corner of Broadway and Duane Streets, an 18-story building, built about five years ago, and also the Post Building, on Vesey Street, a 12-story building; the new Hall of Records directly opposite on Centre Street; the Subway wall abuts and runs parallel to the west building line and its base of rail is at an elevation of about 24 ft. below the street or about 10 ft. above the general excavation. On the east side of the lot is the City Hall Terminal of the Third Avenue Elevated Railroad. The piers for the elevated railroad columns were each carried down to 42 ft. below curb or 10 ft. below the general excavation. On the south end of the lot is the Brooklyn Bridge Terminal, which is perhaps one of the busiest traffic stations in the world. Another feature which gave considerable anxiety and difficulty was the fact that Chambers Street, one of the busiest traffic arteries of the city, carrying horse car tracks, runs through the very center of the lot. Besides the necessity of maintaining the incessant flow of traffic, there were large water and sewer mains and electric light, telegraph and telephone conduits to be kept in perfect condition, with not the least permissible disturbance, as the latter supplies the water and current to a large section of the city. This street was supported on long steel girders resting on timber bents carried on temporary concrete piers. The water and electric service mains were carried on bracket hangers suspended from the girders, as shown on sketch 3. Running along the northerly end of the lot are 150 ft. of 4 by 6 ft. concrete electric

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conduit, and along the westerly end 175 ft. of 36-in. water main under 50-lb. pressure and 160 ft. of 200-lb. pressure fire service water mains carried on a temporary timber trestle. During the

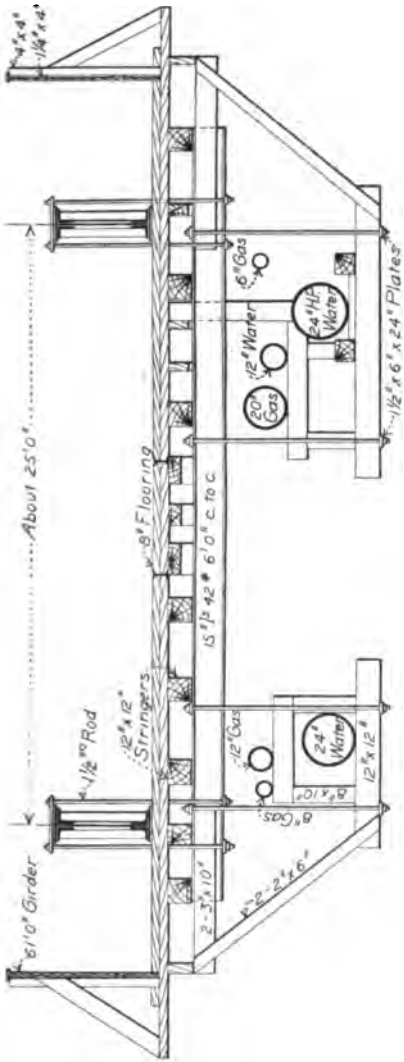

FIG. 3. Section of Bridge Carrying Chambers Street Across the Building Site. Method of Supporting Service Mains.

sinking of the caissons along this bridge, especial care had to be taken to prevent undue settlement of the piers supporting same. The latter was therefore placed on screws and check levels taken daily. If any settlement occurred, no matter how slight, correc-

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tion was made by adjusting these screws. In one case the total settlement was about 3 inches, so that the lower part of the concrete pier pulled away from the structure, leaving the latter entirely supported on these screws. During no part of the work was the traffic or any part of these structures seriously interfered with.

All of these structures interfered greatly with the sinking of caissons near and underneath them. In view of the scant head room, varying from 12 to 20 ft. to the underside of those structures, the build-up of those caissons was limited to that height, and in some cases the air lock could not, as is the general practice, be placed in the center of the caisson but had to be shifted to either end in order to secure a reasonable build-up of concrete for sinking. The working chamber was first built up at the lot level, which required one form 4 ft. in height. Then another 8-ft. form was placed over this. The excavation was then carried on in the working chamber and the caisson ditched about 4 ft., thus permitting an additional 4 ft. form to be placed, giving in all a height of 16 ft. of concrete. The air lock was then put in place at one end of the caisson and the sinking proceeded with. This, of course, increased the cost of sinking, as the height of the concrete build-up is a very important factor of the cost of pneumatic caisson construction.

The general excavation of the lot was carried to 32 ft. below curb. Along the wall of the subway steel sheeting of the corrugated or Wemlinger type was driven to a depth of about 21 ft. below the base of the subway wall or about 9 ft. below the general excavation. On the east side of the lot heavy wooden 3-in. sheeting braced with long 12-in. by 12-in. inclined spurs were used to support Park Row. On the extreme north of the lot was an inclined runway over which was hauled the wagon loads of excavated material to the street. None of the adjoining buildings were found to have settled appreciably. The settlement in the elevated columns was about 2 inches, but being held on screws could be easily adjusted. The maximum settlement along any part of the subway wall did not exceed three fourths of an inch. Considering the fact that some slight settlement is generally to be expected in the soil adjoining the sinking of a pneumatic caisson, and that the vibrations caused by the subway and the elevated structures would tend to accelerate the settlement, it is remarkable that it was not much greater.

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The necessity of carrying the caissons at the south end of the lot to rock to a depth of about 100 to 102 ft. below water level involved to considerable danger the men excavating in the working chambers at that depth. In order to hold back the fine sand and water at such great depths when working in the caisson, it was necessary to introduce therein compressed air at a pressure as high as 45 lbs., or about 3 atmospheres above normal. About .43 lb. of pressure is required for every foot below water level, and as the pressure increases the length of shift is decreased. With the exception of the work at the Eads Bridge at St. Louis years ago, this was probably the highest air pressure ever used on any caisson work. The tunnels under the Hudson and East rivers did not employ more than 40 lbs. There the men worked in three-hour shifts and took 48 minutes for decompression. In coming out of the tunnel at this pressure the men had to walk about 1,000 ft., and passed through three compartments from which the air pressure was stepped down from 40 lbs. to 29 lbs. and 12½ lbs. above normal atmosphere. In the caissons for the Municipal Building at 45 lbs. pressure, the men worked two 40-minute shifts per day and took about 35 minutes for decompression before the men were permitted to pass into the normal atmosphere. The difficulty of ventilation is much greater in caisson work than in tunnel work, as the latter forms a much larger air reservoir, and the air and ground is generally found to be at the same temperature as the river water overhead. In caisson work the space is no larger than a good-sized room and the air temperature much higher. Extraordinary precautions were necessary in operating under these high pressures when decompressing, and after. The men were carefully examined by a competent physician who was continually in attendance. A hospital air lock was located on the lot and proved to be a good preventative of the caisson disease (see Fig. 5). Whenever anyone showed symptoms of "the bends" he was placed at once in this hospital lock and the air pressure brought gradually to that in which the man had just been working in the caisson. This lock was well heated and lighted by electricity, and small lounging platforms were located so that the men could rest during the period of decompression. At intervals during this period of rest they were advised to exercise with small iron dumb bells. It is now generally thought

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that the caisson disease is mainly due to an excess of nitrogen, also to a confinement of air bubbles in the blood, which by the excess pressure on the nerves and by the resultant action on the heart produce the symptoms of caisson disease. To assist the release of these, electrical massage and light exercise is advisable. During the lowering of the air pressure the atmosphere is suddenly chilled, so that unless the men are well clothed there is great likelihood of their contracting serious colds. In some of these caissons there were as many as six men in one working chamber, and on this job there were employed as many as 150 "sandhogs" throughout the twenty-four hours. Nevertheless, not a single fatal case occurred, and the few cases of the bends that did occur were successfully treated. The greatest number employed on this work was about 500 men in twenty-four hours.

An attempt to lower the water level covering the lot was made by sinking a sump caisson to a level of about 70 feet below water level at the north end of the lot. This caisson was to be finally sunk to bed-rock, but was so designed that it could be temporarily used as a sump into which the ground water could collect. The water was then pumped out of same and used in the mixing of concrete and for cooling purposes. But this plan was found to be impracticable, as the water did not flow into the caisson fast enough and the water level on the outside of the lot, in spite of continued pumping, showed no appreciable fall. At the north end of the lot a steel sheet-piled sump was put down from which the ground water was pumped and used for mixing concrete during the work.

The general excavation of the lot, down to a level of about 32 ft. below curb, was carried on with a steam shovel which loaded the spoil directly into wagons, the latter entering and leaving by means of the inclined runway at the north end of the lot. A heavy timber trestle was constructed, one north of the Chambers Street Bridge and the other south of it, on each of which was installed a tower traveller carrying four derrick booms (Fig. 4). These, together with several other stiff leg derricks, were so placed as to command the whole area of the lot and served for unloading the timber caissons from the wagons in the street and for handling all materials. During the months of June and July there were as many as twenty-one booms in operation. The operation of these derricks

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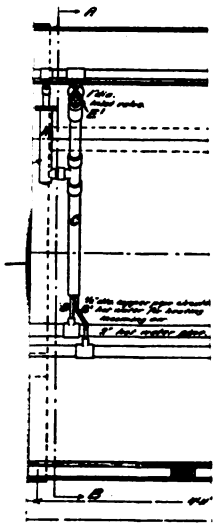

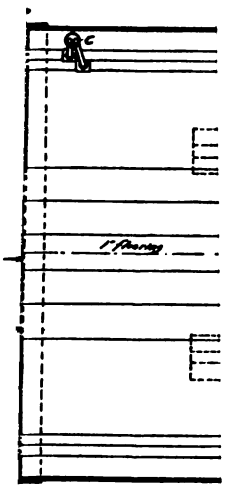

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forms one of the main items of cost on this work. They were used to hoist the completed timber caissons into position on the lot, to set

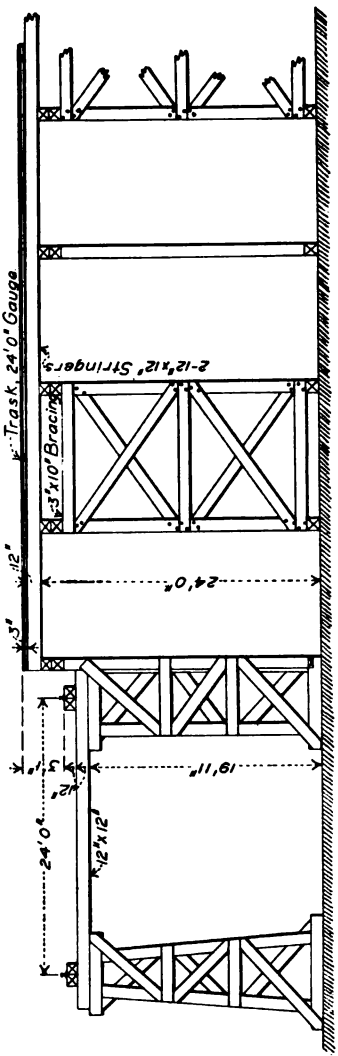

FIG. 4. Temporary Trestle—for Traveling Derrick.

the steel caissons and steel forms for the circular caissons and the wooden forms for the rectangular caissons.

On this job there were four different types of caissons or working chambers used, as shown on sketch, Figs. 6 and 6*A*. Type A shows a circular steel working chamber used only for caissons of a

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diameter between 6 ft. 6 in. by 8 ft. 6 in., the former being the smallest size caisson used on the job. Eighteen of these were used. These were made of riveted steel plates, $\frac{1}{2}$ to $\frac{7}{8}$ in. thick and 78 in. high. The roof of these working chambers consisted of a $\frac{3}{8}$-in. flanged head plate.

Type B is the rectangular timber working chamber that has been generally used on all caisson work for high buildings. The thick-

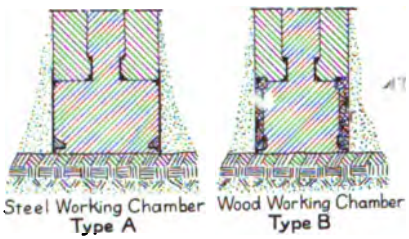

FIG. 6. Typical Caissons.

ness of the timber walls vary from 8 by 12 in. to 12 by 12 in. Ten of these were used on this work and only for caissons of width between 5 ft. 6 in. and 7 ft. 6 in., inclusive. The smallest caisson of

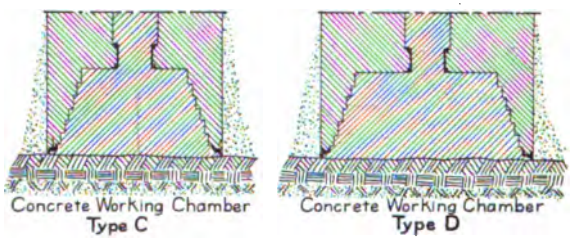

FIG. 6*A*. Typical Caissons.

this type is 6 ft. 6 in. by 11 ft. and the largest 7 ft. by 22 ft. These working chambers had to be built in the contractor's yard at Greenpoint and hauled to the site, where they were raised by the derrick and put in their exact positions.

Type C were the caissons having circular concrete steel working chambers and were used only for those having diameters between 9 ft. and 14 ft. 2 in. The largest caisson of this type is 14 ft. 2 in. in diameter. Circular steel forms were used to build these working chambers and were made of two sections of $\frac{1}{8}$-in. metal stiffened by 3 by 3 in. angles. There were 22 such working chambers built.

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Type D are the rectangular concrete steel working chambers, of which 56 were built in place. This type of concrete steel caisson was used only for rectangular caissons of width greater than 8 ft.

The largest caisson ever sunk for a building is the one in the northeastern part of the lot. It is 26 by 31 ft., carrying 6 columns of a total load of about 5,000 tons.

The headroom inside the working chambers is about 6 ft.

Wooden forms were used for all rectangular caissons and steel for the circular ones.

After the wo'-ing chambers were put into position and the forms set up, the concrete was built up inside same to a height of 12 to 50 ft., depending upon the location, size and depth to which the pier was to be sunk. After the first build-up of concrete the caisson is sunk through the water-bearing ground with the aid of compressed air until the top of this concrete has reached water level, when another section of concrete is built up and sunk, and so on until the piers have reached the required depth. The Chambers Street Bridge and important service mains interfered with the construction of the caisson, so that the height of the build-up of concrete was limited to the amount of head-room below such structure, as mentioned above.

The material excavated from the caissons was hoisted out in cylindrical buckets varying in size from 22 in. to 34 in. in diameter and 33 in. to 46 in. in height and with a maximum capacity of about 1 cu. yd. There were as many as 32 dirt buckets and 21 concrete buckets used at one time. Much of the spoil taken from the caissons was a good coarse sand, which was therefore piled at convenient points on the lot where it was reserved for use in making concrete. The finer sand, which was not suitable for making concrete, was dumped in large elevated storage hoppers. These hoppers have a maximum capacity of about 20 cu. yds. and unload by gravity into the wagons passing under it, and in which the spoil is removed from the lot. These hoppers could be moved about from pier to pier and could gather the spoil from three or four neighboring caissons, but soon became overloaded, so that the sand had to be piled on the lot. They were finally abandoned as it was found at times that the buckets were hoisted from the caissons faster than the hoppers could be emptied, the lot being so

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crowded that the wagons could not get to and from the hoppers quick enough (see Figs. 7 and 7*A*).

The caissons were equipped with patented collapsible shafting,

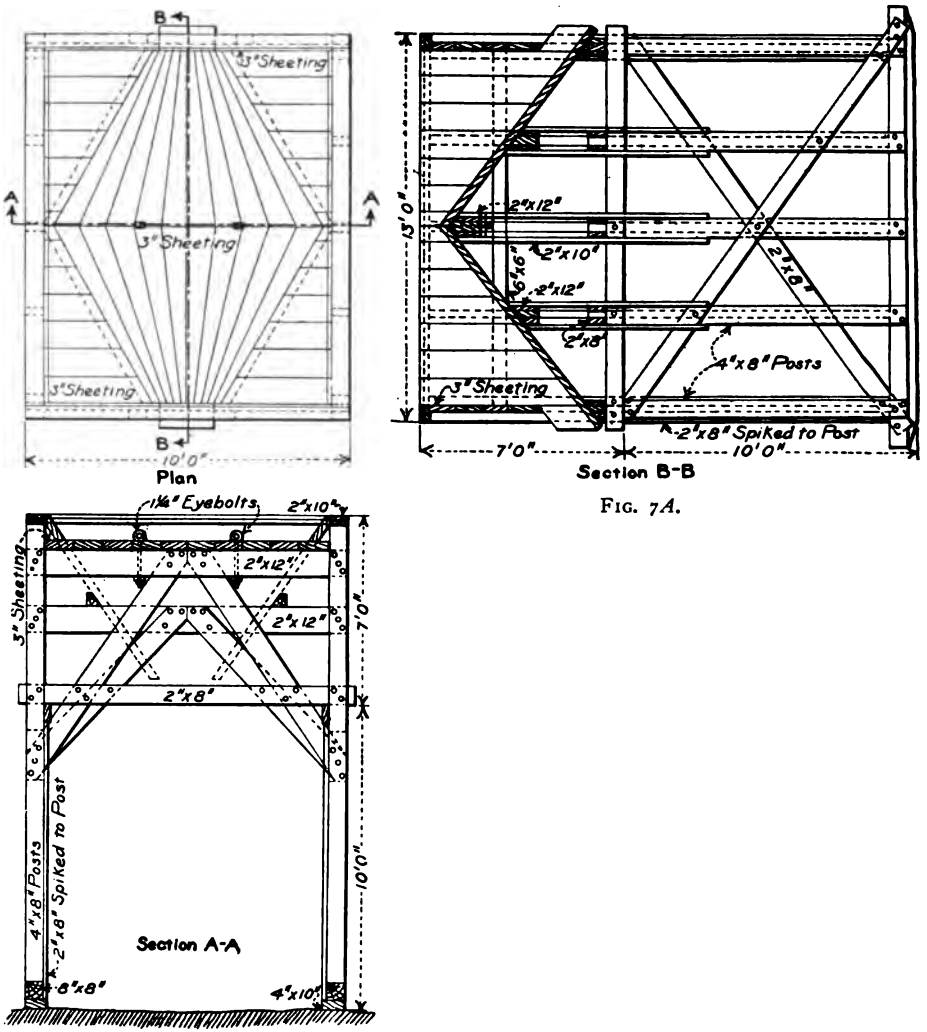

FIG. 7*A*.

FIG. 7. Movable Hopper for Excavated Material.

located generally in the center of the caisson and forming a passage or shaft communicating from the open air to the working chamber below, with separate compartments for men and materials.

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Some of these are oval in shape, but a more improved type, egg-shaped in form, was used on this work for the first time.

Owing to the great depth of material through which some of these caissons had to be sunk, some very great frictional resistances were met with. In order to sink them to the levels required, pig iron blocks weighing from one to one and three quarter tons each were piled on the concrete caisson. As much as 1,000 tons, in addition to its own weight (which reached in cases 1,500 tons), was required to sink some of the caissons to bed-rock. The frictional resistance was computed and found to vary generally from 300 to 500 pounds per square foot, and reached as high as 800 pounds per square foot.

The caissons are filled with 1–2–4 concrete, which is mixed with Lehigh cement in three Rancome and three Smith mixers. These mixers were located under a working platform at street level (Fig. 8) and discharged into bottom dump steel buckets handled by the traveller and derrick booms having a maximum radius of operation of about fifty feet.

The first building in New York City whose foundations were constructed by the aid of the pneumatic caisson process, was the Manhattan Life Building, in 1892. Since then over fifty other skyscrapers in the lower part of Manhattan Island have been built on pneumatic caissons. During these eighteen years many changes have been brought about in the construction details. During the first five years the caissons were generally built of steel and brick; later and to this day the working chambers were constructed of heavy timber walls above which the concrete mass of the pier was built, forming the footing for the main columns of the building. The deep foundation work for the Municipal Building is of special interest in that it marks another important advance in the practical application of the compressed air process to foundation construction. To be sure, each important advance in any branch of engineering is based upon the fundamental principle of economy and safety of construction; so that the improvements developed on this work will go far toward decreasing the cost of pneumatic caisson construction.

For the first time the working chambers were built of reinforced concrete, forming one monolithic mass with the main body of the

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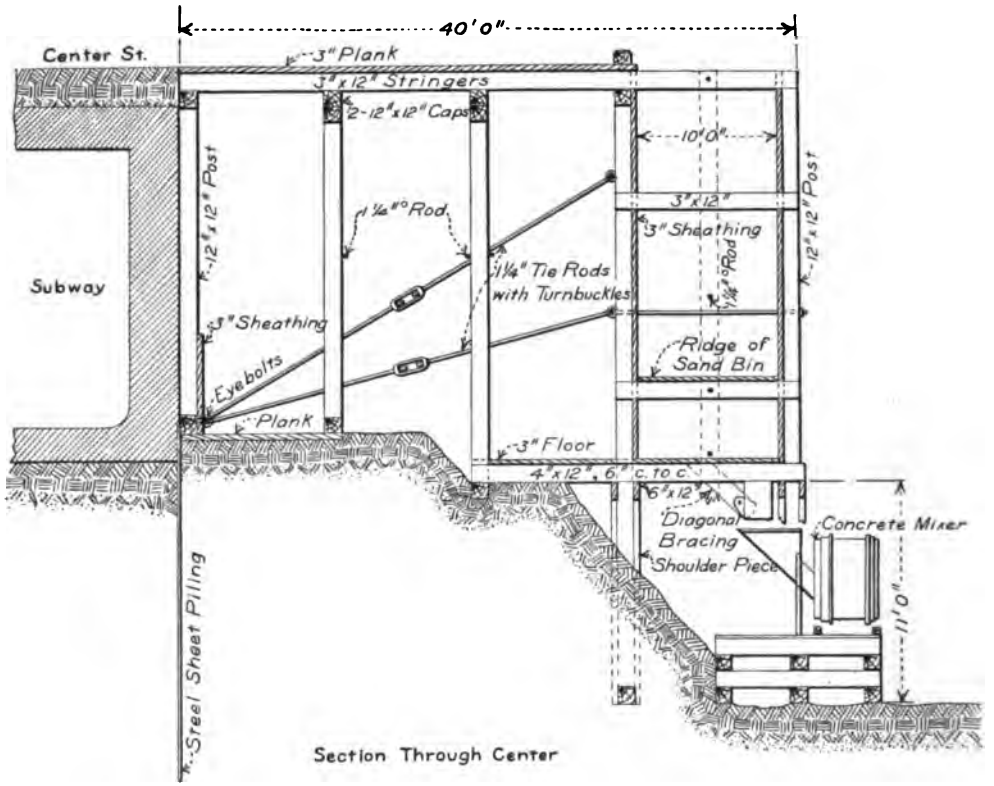

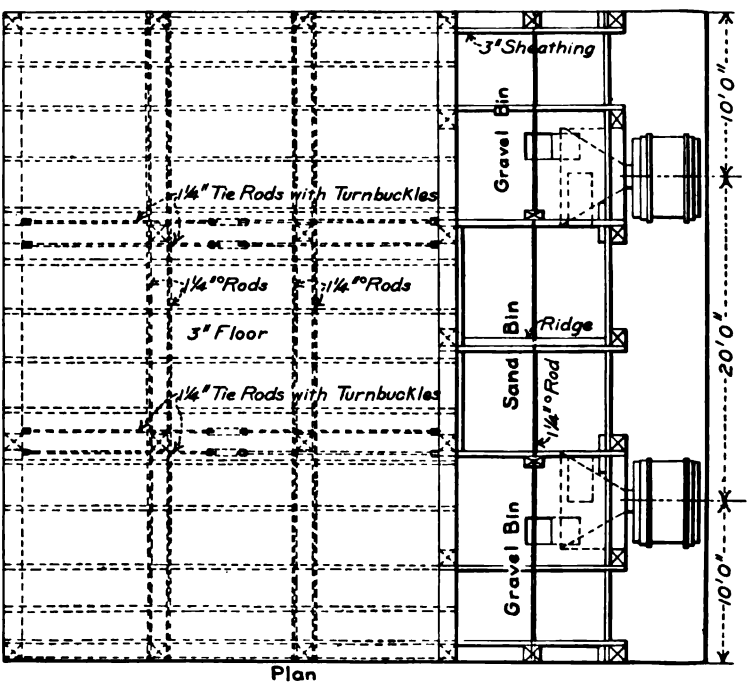

FIG. 8. Concrete Mixing Plant.

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pier and containing no exposed iron or timber. The various types of caissons were adopted where the conditions were most favorable for the use of a particular form. Not only are these more econom-

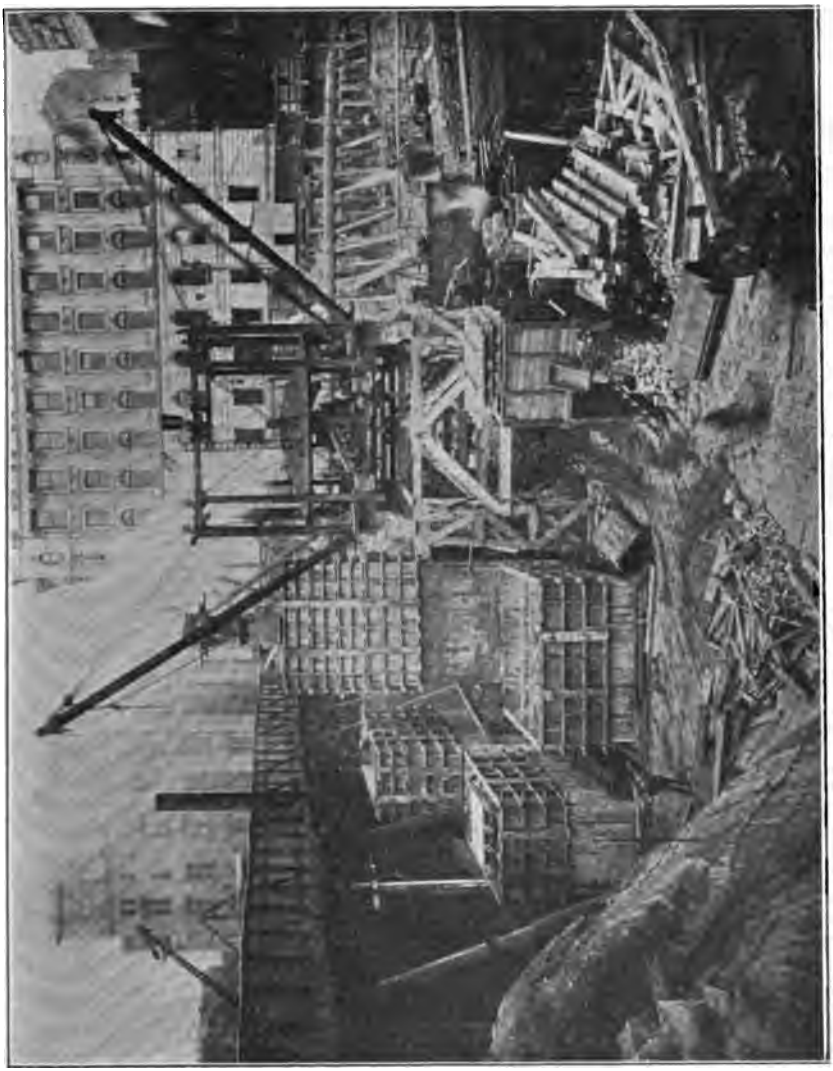

FIG. 9. View of North End of Lot—Traveling Tower Derrick—Forms for Rectangular Caissons Movable Spoil Hopper.

ical in cost of material but save much expense in labor and time of construction. Improvements in the patented shafting leading from the working chamber to the top of the pier in the air-locks, buckets,

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etc., have been made with the idea of decreasing the cost of hoisting the spoil from the working chamber. Aside from this consideration, this work is notable for its magnitude and rapidity of construction under great difficulties; also for the manner in which an unforeseen contractual difficulty was finally adjusted in a manner equitable to both parties. Air-pressure was put on the first caisson on November 23, 1909, and was removed from the last caisson on September 3, 1910. In spite of the delay resulting from adjustment of contract and the cold stormy winter of 1909, the 106 caissons were completed within one year.

The general contractor for the foundation work was The Foundation Company, New York City. This work was under the super-

FIG. 10. Bird's-eye View of Entire Lot Showing Chambers St. Bridge, Three Tower Derricks, Location of Power Plant, Number of Caissons Built Up Ready for Sinking.

vision of the Department of Bridges—Mr. Kingsley L. Martin, commissioner; Alexander Johnson, chief engineer; F. W. Perry, engineer in charge; McKim, Mead & White are the architects.

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Owing to the magnitude of this work and the rapidity with which it had to be carried on, a very large plant was required. The following, which was probably one of the largest equipments ever used on any one job of this kind, was installed by the contractors.

The author wishes to acknowledge with thanks the courtesy of the Engineering Record and Engineering News in granting the privilege of using some of the illustrations printed with this article.

PLANT USED ON MUNICIPAL BUILDING JOB.

BOILERS:

Center St. Battery:

2 Kewanee at	80 H.P.	160	H.P.
1 Beggs,	125 H.P.	125	H.P.
1 Phœnix,	125 H.P.	125	H.P.
1 Erie City,	100 H.P.	100	H.P.
1 Oil City,	100 H.P.	100	H.P.*
1 Nagle,	75 H.P.	75	H.P.
6 Bigelow,	175 H.P.	1,050	H.P.
Total		1,725	H.P.

Following boilers in Chambers St. Battery:

1 Oil City,	100 H.P.	100	H.P.*
1 Nagle,	60 H.P.	60	H.P.

* Same boiler.

COMPRESSORS:

5 Ingersoll-Rand, 22 × 26¼ × 24 in.
1 McKiernan, 22 × 26 × 24 in.
1 Ingersoll-Rand, 18 × 30 × 22 × 22 × 24 in.
3 Kelley Berryman Freed water heaters.
2 Sets twin air receivers.
3 Ingersoll-Rand after collers.
1 McDougal punch and shears.
1 Arminton Simms engine.
1 Williams pipe threading machine.
1 Boynton and plummer bolt threading machine.
1 Medical lock.
1 Ingersoll steam hammer.
5 Ingersoll steam drills.

HOISTING ENGINES:

32 Lidgerwood engines.
3 Lambert engines.
1 Mundy engine.
26 Total engines.

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SWINGING GEARS:

14 Lidgerwood.
12 Mead Morrison.
26 Total.

DERRICKS:

2 (8) Steel Traveler Nos. 34–37; Nos. 41–44.
1 (4) Wooden Traveler Nos. 19–22.
1 Jinnywinch.
1 Stiff leg No. 23.
1 Stiff leg No. 27.
1 Stiff leg No. 28.
1 Stiff leg No. 29.
1 Stiff leg No. 30.
1 Stiff leg No. 33.
1 Stiff leg No. 38.
1 Stiff leg No. 39.
1 Stiff leg No. 40.
1 Stiff leg No. 24.
3 Stiff legs Nos. 25, 26 and 31.
26 Total stiff legs.

PUMPS:

1	Pulsometer,	No. 3574.
1	Smith-Vale,	No. 60603.
1	Cameron,	No. 26216.
1	Cameron,	No. 28319.
1	Worthington,	No. 152933.
1	Worthington,	No. 57397.
1	Deane,	No. 57178.
1	Cameron Sinking Pump,	No. 34072.
1	Cameron Sinking Pump,	No. 33352.
2	Watson-Stillman electric pump.	
1	Brooks electric pump, motor	No. 334470.

LOCKS:

1 Hospital air lock.
5 Air locks for horse-shoe shaft.
7 Air locks for oval shaft.
4 Air locks for 30-in. round shaft.
16 Total.

MIXERS:

2 1 C.Y. Ransome.
3 1 C.Y. Smith.
1 ¾ C.Y. Ransome.
6 Total.

BUCKETS:

1 ½ C.Y. Orange Peel.
4 ½ C.Y. Cyclopean.
8 1 C.Y. Tidewater.

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1 1 C.Y. Doud.
10 1 C.Y. Steubner dirt.
22 Caisson dirt buckets.
10 Caisson concrete buckets.
56 Total.

MISCELLANEOUS:
1,600 one-ton C.I. weights.
500 1¼-ton C.I. weights.

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THE CHARACTER OF THE HUDSON GORGE AT NEW YORK CITY.

By G. S. ROGERS.

The extraordinary number of great engineering enterprises connected with New York City has enabled us to subject most of its natural features to an unusually minute scientific scrutiny; but it is only recently that the rocky bed of the Hudson River has become susceptible of study. Of the depth and character of the loose materials which partly fill the old channel we have been comparatively ignorant, although their importance as a means of determining the amount of elevation which caused the excavation of the old channel has been recognized, as well as their great value to engineers. About 1907 the first borings were taken for the Hudson crossing of the new city aqueduct, in the vicinity of Storm King Mt., and about the same time borings were made by the Pennsylvania Railroad for its crossing at 32d St., N. Y. Tunnels have also been put through by the Hudson and Manhattan Tunnel Company at Cortlandt St., about three miles south of the Pennsylvania. The data obtained by these three explorations have been made public rather recently, and so far as I am aware only the first has been written of. The Harlem and East rivers are more familiar, owing to the gas tunnels, the Rapid Transit tunnels, the numerous bridges, etc. Finally, the submerged channel of the Hudson has been traced out of the bay to the edge of the continental shelf by means of soundings which have accumulated for the last fifty years, and which are, of course, very numerous and exact owing to their value to navigators. It is with the facts observed at these localities, but especially with those at the Hudson River tunnels at New York City, then, that this paper is concerned.

The writer desires to acknowledge his indebtedness to Professor J. F. Kemp for his kind aid in collecting much of the data given herein, and to Professor C. P. Berkey for the many valuable suggestions which his intimate knowledge of the geology of the region affords.

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The general topographic features of the Hudson Valley are so well known that a word concerning them will suffice. The Highlands form the dissected remnant of the ancient Schooley peneplain, which was the product of a Cretaceous cycle of erosion. Its reëlevation to undergo the Tertiary cycle is attested by a series of rocky shelves along the banks, at an altitude ranging from 140 to 350 feet, and representing the traces of the ancient valley.* Finally, this broader valley having been formed, drainage was again revived; and it was during this period, immediately preceding the inauguration of the ice age, that the river eroded the narrower channel of whose details we are just now becoming cognizant. The continental glacier then ploughed along the river bed, and on melting filled this channel with till; and the considerable thickness of fine mud and silt which are found to-day overlying this till are what would be expected from the general depression of the land subsequent to, or during, the retreat of the glacier. A subsequent elevation, however, must then be postulated to account for the extensive erosion of these estuary deposits, and of the terminal moraine at the mouth of the river (The Narrows) through which there is a gap of 240 feet deep and one mile wide. Finally, the depression followed which is apparently still progressing, leaving the river shallower, shorter and narrower than of old.

The geology of the banks of the Hudson is also somewhat complicated. To the north are found the hard slates, sandstones and limestones of the Ordovician and Cambrian. Further down in the Highlands lie the even more resistant and exceedingly complex metamorphics, and just below the plutonics of the Cortlandt series. Other metamorphics, generally bound together by many pegmatite dikes, continue down the east side of the river to New York, while opposite the city on the New Jersey side we find the softer arkose sandstone of the Newark series, with the intruded sheet of Palisade diabase outcropping a short distance above present water level. Still further south, on Staten Island at the Hudson's mouth, the river flows over the soft incoherent Mesozoic and Tertiary, while below this (near Sandy Hook) the old submerged channel gradually becomes free of sediments and susceptible of close determination. The rather remarkable straightness of the river at New York

* See for further details, J. B. Woodworth, "Ancient Water Levels of the Champlain-Hudson Valleys," Bull. 84, N. Y. State Museum.

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has been variously explained by various authors; the data at our command, however, are insufficient either to disprove or to sustain any of these speculations. Thus also in the Highlands there is considerable difference of opinion, but proof is beyond us at the present time.

Crossing Near Storm King.

The data utilized in this connection were gathered by the Board of Water Supply of New York in their explorations for the new aqueduct. The Storm King crossing has been written of by A. D. Flinn,* and data along the whole line of the acqueduct, with especial reference to the crossings of the Hudson and its tributaries have been published by J. F. Kemp.†

Of the six crossings of the Hudson which have been explored by the Water Board, only those at Pegg's Pt. and Storm King need be considered. At Pegg's Pt. (Fig. 1), ten miles north of Storm King, two cores were taken; the deepest drill caught slate at 223 ft. and the other struck limestone at 92 ft. In view of the remarkable depth at Storm King we are forced to conclude that a gorge of two or three hundred feet exists somewhere in the 1,040 feet between these two holes.

The Storm King crossing itself is located in the Highlands. Storm King Mt. on the west and Breakneck on the east are only 2,800 ft. apart, their granite heights rising abruptly to over a thousand feet. Detailed diamond drill exploration has been prosecuted at this point, but owing to two heavy layers of boulders only three of the eight more important drill holes reached bed rock. Hole 16‡ at 300 ft. from the eastern bank caught the granite bed-rock at 200 ft. Hole 19, at 560 ft. out, stopped in the second bed of boulders at 419 ft., but probably near bed-rock. Hole 22 caught granite at 507 ft. Hole 10 is about 300 ft. south of the last, out of the direct line, and is the most significant of all. It revealed granite identical with that on the banks at 608 ft. and penetrated it for 8.8 ft. Hole 20 struck a very heavy stratum of boulders at

* "Explorations for Hudson River Crossing of the Catskill Aqueduct, New York City," *Engineering News*, April 2, p. 358, 1908.

† "Buried Channels beneath the Hudson and its Tributaries," *Amer. Jour. of Science*, XXVI, 301, 1908.

‡ Compare map (Fig. 2) which, with the data used in this section, was taken from Professor Kemp's paper.

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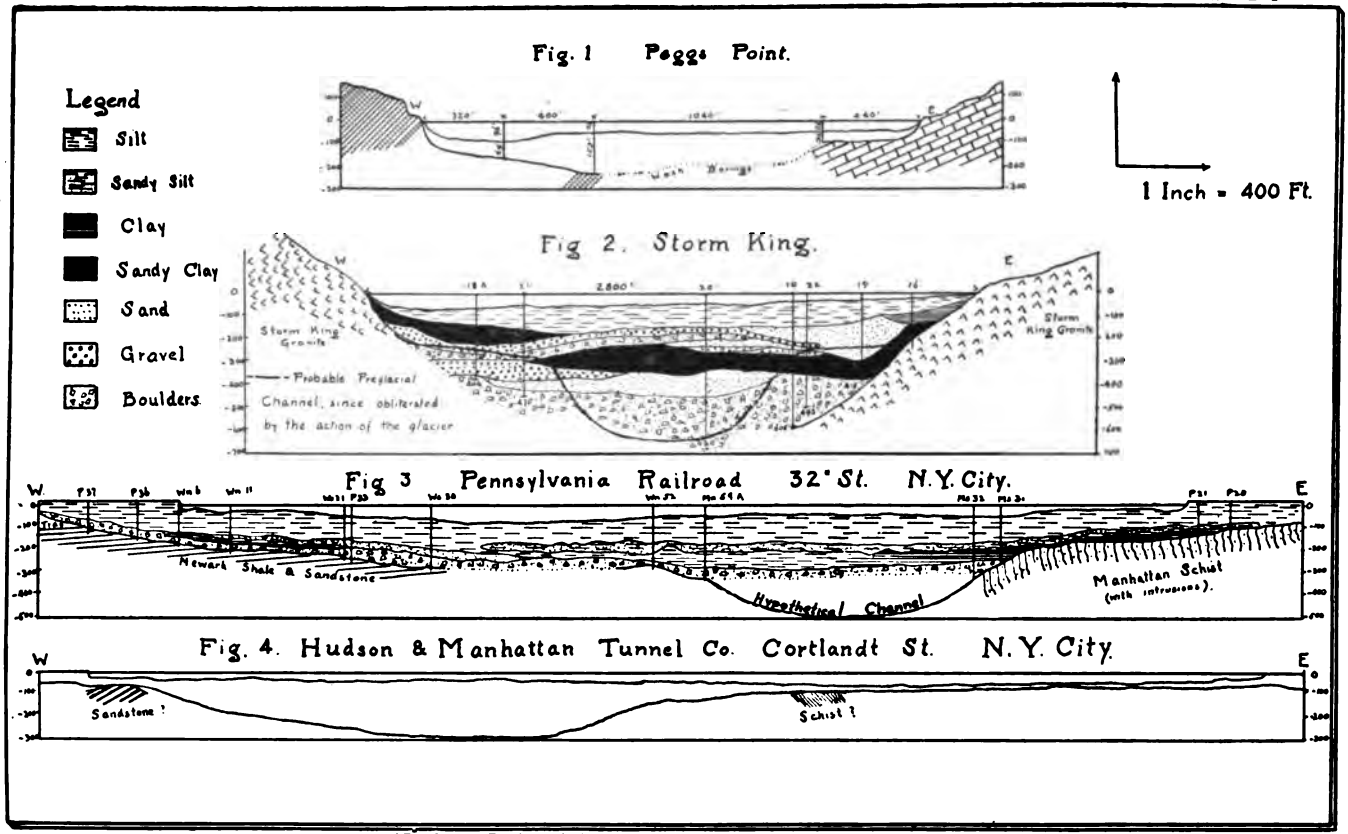

(Scale as Reduced for Publication is 1 inch = 800 Feet.)

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580 ft. and was temporarily stopped but has since been driven to 630 ft. and has stopped again in boulders. Hole 21 at 700 ft. from the west bank stopped in boulders at 475 ft. Hole 18 at 500 ft. from the west bank was probably near bed-rock at 404 ft. if not in it. Thus the critical hole appears to be number 20, but at the present time it can only be said that the river is deeper than 630 ft. at this point.* The ancient river, therefore, was here narrow and deep, at least 630 ft., but probably not very much more, since although this hole is not in the center of the stream, the glacier operated to flatten the bottom and give it a U-shaped section.

Pennsylvania Railroad Tunnels, 32d St., New York City.

In the following description the private profile maps of the Pennsylvania Railroad have been the chief source of information. It will be found here given, reduced to a natural scale. Microscopic examination of the drill cores, which were also furnished by the company, have afforded such petrographic descriptions as are given.

We are struck on first examination of the appended profile map by a rather startling contradiction to the ideas obtained at Storm King. The river at 32d St., New York City, is 4,450 ft. wide at the present time, and was about 6,500 ft. in pre-glacial times; but the peculiar thing about this profile is that it shows a depth to bed-rock of only 320 ft. as against over 630 ft. at Storm King. Moreover, it is noticeable that the layer of glacial, water-bearing gravel attains a maximum thickness of only about 80 ft. as against about 200 ft. at Storm King.

The explorations made by the railroad have been quite complete if wash borings are included: 80 borings in all have been made, 26 in 1901 before tunneling, and 54 in 1906–1908, driven from within the tunnel itself. Thus the borings average less than 90 ft. apart, but of the whole number only about twenty are diamond drill cores. Of these again only seven are actually underneath the river—the rest are driven from the banks, and of course penetrate the Newark series on the one side and Manhattan schist on the other. Finally, these seven cores are so distributed that two are about 1,200 ft. apart and two others 1,000 ft., so that the list of data is somewhat fragmentary.

* Since date of writing another hole has been driven, which stopped in boulders at 751 ft. The river is therefore at least 751 ft. deep at this point.

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Before taking up the locations, etc., of the diamond drill holes, a word about the material above bed-rock may not be out of place. As shown in the maps, silt forms the river bottom to a depth of from 100 to 150 ft. It is, of course, more or less mixed with clay and mud, but is on the whole quite homogeneous. Its hardness or resistance to boring varies, but increases with depth. It is dark gray, and carries 34–40 per cent. of water; but its fineness is its most striking character, 83 per cent. passing through a 200-mesh sieve. Below the silt a huge lense of a sandier variety was met, averaging about 60 ft. in thickness and extending west about 3,480 ft. from a point 320 ft. east of the east bank.* This material contains 30 per cent. of a fine gritty sand, and carries water and an inflammable gas. It is noteworthy that there is usually a thin crust between the "silt" and this "sandy silt." Below this lense and in communication around the ends of it with the silt above is a more compact variety, evidently the same except as hardened and dried by the greater pressure. To the east this grades into a blue clay, and to the west there is a more sharply defined lense of red clay and another of gray sand and gravel. Finally, underlying all (except for a distance of 880 ft. from the east bank) glacial gravel or hardpan was met. This consists of subangular stones of all sizes from pebbles to large boulders, and causes considerable difficulty in boring. It carries salt water under a head equal to that at the surface of the river, and must therefore be in free communication with it at some point. As mentioned above, it seldom reaches 80 ft. in thickness, and often dies away to 15 or 20.

As shown by the drill cores, the Newark arkose extends at least 1,100 ft. from the west bank, hole Ws 30 being the last to penetrate it. In hole P 38, near the foot of the Palisades, a sheet of the ordinary Newark diabase was also met, intruded between the beds of slate. Between this sheet and that which forms the Palisades ridge the slates are pretty thoroughly baked; but below it the rock becomes coarser and less susceptible to contact effects. The sheet of diabase lies 68 ft. below the surface, and is itself about 80 ft. thick. Its dip is presumably that of the beds at the surface—about 13° N.W. The cores show a perceptible tendency in

* References to the "bank" are with regard to the present bank of the river—not the ancient rocky bank.

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the arkose to increase in coarseness with distance from the west bank—*i. e.*, as successively lower beds are penetrated—until in hole Ws 30 the rock is coarser than ordinary mortar, the grains averaging over 1.5–2 mm. Also the rock exposed by these holes differs from that at higher levels in being generally unoxidized—the characteristic red or black color is usually lacking, and the rock is practically white.

Hole Ws 30 caught the rock at 270 ft. and penetrated it for 44 ft., bringing up a fairly coarse white arkose sandstone, the grains averaging 1.5–2 mm. and sometimes reaching 6–8 mm. Some of the feldspar retains its pink color, but most of it is kaolinized enough to be a chalky white. The quartz makes up about half the bulk of the rock, but the grains are fairly even in size (1–2 mm.), only a few attaining the dimensions of the larger feldspars. Rarely a flake of muscovite is seen. The rock in general appearance resembles a granite, and was so named by the engneers. Microscopic examination confirms this description. The quartz shows considerable secondary enlargement, and strain is indicated by the numerous rehealed cracks and by wavy extinction. The orthoclase, which with the quartz makes up the great bulk of the rock, is always more or less altered, but decomposition has not proceeded very far. There is a large amount of interstitial material, chiefly kaolin, which was probably deposited as clay between the sand and feldspar grains.

The next hole—Wn 52—lies 2,080 ft. from the west bank and reveals at 260 ft. a very peculiar and complex rock. Megascopically it resembles Lake Superior jaspilite, being apparently chiefly iron-stained quartz, penetrated by fine white quartz veins. Under the microscope the most striking feature is the line structure in which the very fine material is arranged surrounding little rounded grains of quartz. The general appearance of the rock points to a metamorphic origin, these lines representing a shear structure. The rock has evidently been indurated, for the great bulk of it is quartz; but while there is apparently much secondary quartz, grains which appear to be of clastic origin are also plentiful. Under crossed nicols the recrystallization becomes apparent; fairly large angular patches of quartz are then seen, arranged without regard to the general structure of the rock. There is practically no feld-

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spar; but there is a considerable amount of a fibrous material, somewhat radially arranged, with parallel extinction; probably a secondary amphibole. Tiny flakes of hematite are very plentiful along the shear lines, and furnish one of the indications of the metamorphic character of the rock. This hematite is presumably derived by metamorphism from the fine reddish, ferric material which is so extremely plentiful in the rock and which may be very finely divided hematite itself. Some magnetite may also be present, but the fine black grains, wherever determinable, appear to be hematite. Tourmaline is sometimes also seen. The origin of this rock is so obscure that nothing can be said definitely about it. It is evident that it has suffered shearing, recrystallization, and lastly crushing, and that it was originally a fairly quartzose rock. It is unlike either the Manhattan schist or the Newark sandstones, although it may possibly represent a layer of fine quartz sand, deposited perhaps by eolian agencies immediately under the Newark. Speculations regarding its ulterior origin are, however, so hypothetical as to be almost fruitless. The complexity of the rock precludes any satisfactory name; perhaps the term sheared iron-bearing quartzite would give some idea of its character.

The next hole—Mn 59A—is just east of the state line, being 2,140 ft. from the east bank. This is the deepest hole of all, revealing bed-rock at 301 ft. and penetrating it for 28 ft. This rock is a fine laminated serpentine, traversed by veins of a highly magnesian dolomite. Under the microscope it resembles the complex mixture of alteration products found at Castle Pt., Hoboken. The rock is chiefly antigorite and chrysotile, with dolomite veinings. There is also considerable secondary amphibole, and the usual amount of magnetite and chromite. There is no doubt that the rock was originally a basic dike or sheet, similar to those which have given rise to the Hoboken serpentine and the outcrops on Manhattan Island, and probably related to them.

Between this hole and the next there is a stretch of 1,200 ft., the latter, Ms 32, being 940 ft. from the east bank. It caught hornblende schist at 287 ft., penetrating it for 30 ft. This schist is similar to that found in various places on Manhattan Island in the Manhattan schist, and is therefore probably a metamorphosed

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dike—gabbro, according to Julien.* The hornblende is the common, dark green, pleochroic variety, but with it is associated a lighter green, also pleochroic material, with a much finer cleavage. The incipient decomposition to chlorite apparent in the hand specimen bears out the idea that this is chlorite, although its cleavage might be taken to indicate rather that it is a green mica. If chlorite, it is probably clinochlore. Quartz, orthoclase, some plagioclase and magnetite are the other constituents, except for an occasional rounded grain of apatite. The other minerals are fairly angular in shape. This rock may be ascribed with confidence to an original basic dike (probably diabase or gabbro), later in age than that revealed in the last hole and therefore less altered.

Hole Ms 31 is only 120 ft. east of this, although it reveals bed-rock at 245 ft. This gives a slope of one foot in three. This rock was only penerated about 7 ft., but since we would expect in this place either Manhattan schist or an intrusion in it, and since its character—it is a very fine-grained granite—would indicate that it is an intruded mass, there is no great doubt but that it represents true bed-rock. Dark minerals are entirely lacking, the rock being made up of quartz, orthoclase, plagioclase, microcline, muscovite and some apatite. The quartz grains are generally secondarily enlarged.

The next core recorded, and the last of interest, since it marks definitely the presence of Manhattan schist, is P 21, which is located 40 ft. east of the bank. It caught the schist at 120 ft. and entered it to a depth of 20 ft. East of this, under the present bank, we find a regular decrease of depth as indicated in the profile. The rocks revealed are either mica schist or granitic material intruded into it.

It appears, therefore, that the evidence at this point is much of the same character as that at the Pegg's Pt. crossing, near Storm King, although at first glance it appears very complete. According to the views of the engineers of the Pennsylvania Railroad, the rocky bed slopes gradually to a maximum depth of about 300 ft. in the exact center of the river, but as a matter of fact a much

* A. A. Julien, "The Occlusion of Igneous Rock within Metamorphic Schist, as illustrated on and near Manhattan Island, New York," *Annals N. Y. Acad. Sci.*, Vol. XVI., Part III., 1906.

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deeper channel may exist on either side of the two holes at this point, so far as the actual diamond drill evidence goes.

Crossing of the Hudson and Manhattan Tunnel Company at Cortlandt Street.

Two tubes have been driven underneath the river at this locality, one at Cortlandt St. (South Tunnel) and the other about 200 ft. to the north at Fulton St. (North Tunnel). In addition to these the same company has also driven a tunnel at Christopher St., about three quarters of a mile to the north. The profiles of the two tunnels near Cortlandt St. have been obtained from the company through Dr. Hovey, of the American Museum of Natural History, but no drill cores appear to be available. As will be pointed out, judging from the character of the profiles, no diamond drilling was done, and we are consequently thrown back on the unsatisfactory and entirely inadequate evidence afforded by wash boring.

The profiles of the two tunnels at Cortlandt and Fulton Streets are so similar that only that of the South Tunnel (Cortlandt St.) is here given. The width of the river is greater than at 32d St., being 5,200 ft. The greatest depth obtained at this crossing is 285 ft., while at Fulton St., 200 ft. up the river, the maximum is 278 ft. While this, of course, is possible, it is entirely improbable that a river flowing over a fairly homogeneous rock should fall 7 ft. in a distance of 200; and when we consider the 301 ft. obtained at 32d St., about three miles to the north, and the 636 ft. at Storm King, roughly 50 miles north, it is evident that the data at this point are unreliable. The point at which the maximum depth, 285 ft., was obtained, was some 2,000 ft. from the west shore, and it will be seen on comparison with the 32d St. profile that at this point the glacial gravel was met; it would seem to be this, therefore, that stopped the drill at this depth at Cortlandt St. (301 ft.). On the other hand, the greatest depth at 32d St. corresponds to a depth of about 100 ft. at Cortlandt St. This would indicate either change in the level of the gravel, or, as is more probable, a swinging to the west of the deepest part of the river in the southerly crossing.

The most significant feature of both the lower crossings is the fact that in projecting the tunnel itself, bed-rock was met at depths of 80 ft. on the eastern side and 60 ft. on the western, at distances

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of 2,000 ft. out in the former case and 160 ft. in the latter. On both sides it only extended toward the center of the river for a short distance, apparently soon shelving off. At 32d St. bed-rock at this depth lies proportionately 400 ft. further west on the western side, and almost 2,300 ft. further east on the eastern, so that apparently the channel has narrowed at this point some 2,700 ft., being only 3,000 ft. wide instead of 5,800 ft. It shows, therefore, a nearer approximation of its cross-section at Storm King, and this rather rapid change in shape has a bearing on the underlying problem which will be emphasized later.

Further discussions of the Cortlandt St. profile would be fruitless; enough has been said to show the unreliability of the data, the only trustworthy fact being the actual encounter with probable bed-rock at a rather surprising distance from the east bank, indicating a marked constriction of the channel. It is thus especially to be regretted that this locality was not properly explored; the only definite fact obtained was a rather surprising one, and would augur well for future developments of a similar character.

Other Records Along the City Waterfront.

The only other records of the depth of the Hudson have been published by W. H. Hobbs* and C. P. Berkey.† These, however, appear to be mainly wash borings, and are therefore practically valueless in this connection, and such diamond drill borings as are given are not in critical positions. The transverse section of the island given on page 25 (Hobbs), however, brings out the fact that the steepness of the eastern rocky bank shown in the 32d St. profile is not a local feature, but that it extends northward at least as far as the fault at Manhattanville.

Harlem and East Rivers.

It will be instrucitve to note briefly, by way of comparison, the characteristics of a few representative sections of these adjacent streams. Owing to the remarkable concentration of engineering

* "The Configuration of the Rocky Floor of Greater New York," Bull. 270 U. S. G. S.

† "Areal and Structural Geology of Southern Manhattan Island," *Ann. N. Y. Acad. Sci.*, XIX., No. 11, Part II., p. 247.

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enterprises along their banks, an unusually complete and accurate set of sections is extant, and these have been brought together by Hobbs.‡ Of them only five are here given, selected as types, and for their completeness and their distribution. The first three are across the Harlem and the other two across the East River. The sections are drawn to scale, and explain themselves.

By way of generalization, it may be said that the rocky channel of the Harlem approximates 1,000 ft. in width and 125 in depth, while that of the East River varies from 2,000 to 3,500 ft. in width, and averages about 200 in depth. The amount of silt deposited appears to be roughly proportional to that in the Hudson, except in the case of Hell Gate, where it has been largely removed by the tidal currents.

It is interesting to note also that with very few exceptions (the most notable being the Gas Tunnel crossing of the East River), the profiles given by Hobbs do not follow the bed-rock continuously across the river, but show a break when it sinks to an indeterminate depth. In some sections also no bed-rock at all is shown; but the feature noted above, whether a meaningless coincidence or not, occurs in ten sections out of the thirteen given of the Harlem in Bull. 270, and in two of the remaining no bed-rock at all is given. In the East River the notable exception to this occurs; but below it, in the 32d St. section, we find somewhat the same feature. In some of the sections this sudden sinking is amply explained by faulting, but in others (notably the High Bridge section) it may be due to erosion, and suggests that a parallel may be found in the Harlem and East rivers to the hypothetical buried channel of the Hudson.

The Submerged or "Hudsonian" Channel.

The evidence regarding this extension of the Hudson gorge over the continental shelf has a debatable bearing on the main subject of this paper. If the gorge be due to river erosion, it enters largely into our conclusions, but if it owes its origin to a current along the sea-bottom, as some think, it is evident that it has little relevancy. It may, however, be provisionally referred to as the submarine extension of the Hudson gorge proper, especially as the bottom of this gorge is below sea level from the head of tide, near Troy, so

‡ *Op. cit.* Also *Bull. Geol. Soc. Amer.*, XVI., 151.

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that the river is really a fiord from that point. The following brief statement of the character of the submerged canyon is taken from J. W. Spencer,* whose work supplements and in a measure includes the earlier investigations of Lindenkohl.† There extends seaward from Sandy Hook a channel, which at a point 93 miles out is 42 ft. deep. At this point it plunges into a gorge, which rapidly deepens by abrupt steps until it attains a depth of 9,000 ft. below the neighboring sea-bottom at a point 71 miles from its head. Just beyond the 3,800 ft. line a lateral gorge enters, according to Spencer, an ancient tributary stream. A short distance below this the gorge widens into a valley, and then opens into an embankment, which receives also the submerged valley of the Connecticut some forty miles to the northeast.

A noteworthy feature of the gorge, in our connection, is its distinctly double character. Spencer states that "a second deeper and more sinuous gorge traverses the outer." Here, then, much more distinctly than in the East or Harlem rivers a second buried channel is evidenced; and if this gorge is really the extension of the Hudson the importance of a buried channel at this point is apparent.

Among the several theories advanced to account for this gorge only two appear to be worthy of serious consideration: the older view, upheld by Dana, Lindenkohl and Spencer ascribes it to river erosion, while the newer, advanced by Forel and Buchanan and apparently favored by Suess,‡ lays the emphasis on submarine currents. The first theory is of course the most obvious, the only objection being the immense amount of elevation required to bring the bottom of the gorge (9,000 ft.) above grade. Spencer, however, after investigating similar phenomena in the West Indies, and enjoying the authoritative corroboration of Hull in Great Britain and Nansen in Norway, feels so confident of the ease with which this 9,000-ft. elevation could be brought about that he declines to allow for any extra subsequent warping of the continental slope, although, on the other hand, he admits that the mountains

* "The Submarine Great Canyon of the Hudson River," *Amer. Jour. Sci.* (4), XIX., 1905, 1–15.

† "Geology of the Sea Bottom in the Approaches to N. Y. Bay," *Amer. Jour. Sci.*, XXIX., 475, 1885.

‡ "La Face de la Terre," v. 2, 1900, pp. 853–856, containing also references to the original papers by Forel, Buchanan, Davidson, etc.

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were then probably much lower than at present. He places the amount of elevation as 9,000 ft., and moreover from a study of the dredgings determines the period of this event as coincident with the early Pleistocene. His chief argument against the second theory, aside from the fact that it is superfluous, is quoted from Professor Shaler: "Subterranean channels must be formed above the base level of erosion, and the establishment of such must precede that of submarine rivers, which soon lose their effectiveness." He also points out, as refutation to objections to the peculiar topography of the submerged channel and gorge, how remarkably it resembles the Niagara River to-day, differing only in greater length and depth.

The second theory* for the origin of this gorge is derived by analogy from that used to explain the submarine gorge of the Congo, and another found in the Gulf of Cape Breton, off the Côte des Landes, France. These cases have been ascribed by their most careful students to the presence of currents which have kept them open, *i. e.*, they are merely unfilled portions of the coastal plain. From the gorges which traverse the deltas of the Rhone in Lake Geneva and the Rhine in Lake Constance, it appears probable that this mode of origin is possible in some cases at least. Yet, however clear this case may be it does not prove the same origin for the Hudson gorge: it appears that Spencer's views should be given more weight in this connection, inasmuch as he has studied not only the Hudson but other rivers in the same territory, and has arrived at similar results in each case.

Discussion.

There are then three direct sources of information as to the configuration of the rocky channel of the Hudson. At Storm King Mt., where the river is narrowest and the bed-rock hardest, a depth of 630 ft. has been attained without striking the bottom of the channel. At 32d St., New York City, the river is much wider and the rock on the west side at least is much softer; at this point the greatest distance penetrated before striking bed-rock was only 301

* See for a somewhat fuller discussion J. B. Woodworth, "Ancient Water Levels of the Hudson and Champlain Valleys," Bull. 84, N. Y. State Museum, pp. 71–2, 1905.

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ft., although the exploration was not complete. At Cortlandt St. a depth of 285 ft. was the maximum attained, but lack of drill cores tends to discredit this evidence. Besides these direct sources there are two indirect; the cross-sections of the East and Harlem rivers and the existence of the double Hudsonian gorge.

The question as to the existence of a deeper buried channel in the Hudson therefore depends chiefly on whether the deep channel at Storm King is the result of normal erosion or whether it is due to extraneous and local causes. A pot hole might be suggested, but it is inconceivable that a pot hole should form in the hardest rock along the whole river. A graben or faulted block would be much more probable, but no evidence of faulting has been found. Again the view has been advanced that the Hudson at one time occupied the Hackensack valley, and a reversal of drainage to the opposite direction might also be thought of; but in the light of geological evidence both of these suggestions are very improbable. Finally, it has been suggested that continental warping subsequent to the formation of the channel might lower the general Storm King area without affecting New York City, or perhaps that an axis exists somewhere between which would even raise the latter. Since, however, we find the rocky terraces regularly rising with respect to sea level as we ascend the river, it is practically impossible that such warping has taken place. The strong probability is, therefore, that the channel is due to erosion, either by water or by ice.

Glacial erosion appears to some to be amply sufficient to explain the depth of the channel at this point. It is certain at least that the channel was greatly flattened and changed to a U shape, and also that its identity as a double channel was destroyed. The action of the glacier, of course, would be accentuated at this point, since the Highlands would constitute the first obstruction to its passage. We have, moreover, all along the river the evidence of the hanging valleys of the tributaries, which according to Kemp were 500 ft. or more above the bottom of the main stream. This indicates strong glaciation; but it must be remembered that the Hudson, during the period of elevation, doubtless cut much more rapidly than its tributaries, and this would in part explain the hanging valleys. Moreover, they would in any case indicate merely strong glaciation

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and gouging all along the river; they do not of themselves point to any concentration of effects at Storm King, and their relevance therefore is less than at first appears. There is left only the general proposition that glaciation everywhere was strong, and particularly so at Storm King; but, as was pointed out above, a glacier tends to plane and flatten its bed and to grind away its irregularities, rather than to actually deepen it. Glacial geologists indeed are divided, some maintaining that it practically lacks vertical erosion power.* It appears wise, however, in the present state of our knowledge, to allow something for glacial action; but 130 ft. at the most would seem to quite cover it. This leaves a channel of 500 ft. (as shown in Fig. 2), due to normal river erosion, which is in turn dependent on land elevation.

The question then arises as to where this channel is at the other crossings considered above. Granted that the depth at Storm King is due to normal erosion, it is very improbable that it should not exist elsewhere. Penck cites only one case of a river's being deeper above than at the mouth; and there (in the Danube) it is due to the differential erosion of a soft rock.

At Pegg's Point, ten miles north of Storm King, there is an unexplored space of 1,040 ft., between holes showing 223 ft. on one side and 92 on the other. If the channel does exist at Storm King it must be located here in this space; if it does not, the river fell 400 ft. in ten miles, which is very improbable. Some traces of this buried gorge are even shown by the wash borings (compare Fig. 1), which indicate a rather sharp drop just outside of the 92-ft. hole. It would be expected, of course, that the double channel complete would be here found, although the two channels had been merged into one at Storm King; for here the rock is much softer and the banks lower, and the glacial action on the inner channel would be therefore considerably ameliorated. There is therefore a negative corroboration of the deep gorge, although a positive is hardly needed at such a distance from Storm King. In the same way at the Pennsylvania crossing there are two unexplored spaces, one of 1,000 and the other of 1,200 ft., the deeper channel being probably in the latter. As is seen in Fig. 2, this does not violate our notions concerning the proportions of gorges;

* See for example H. L. Fairchild, "Ice Erosion Theory a Fallacy," *Bull. Geol. Sci. Amer.*, XVI., p. 13, 1905.

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PLATE II.

Fig 5. High Bridge Harlem River.
1 Inch = 200 Ft.

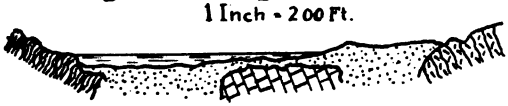
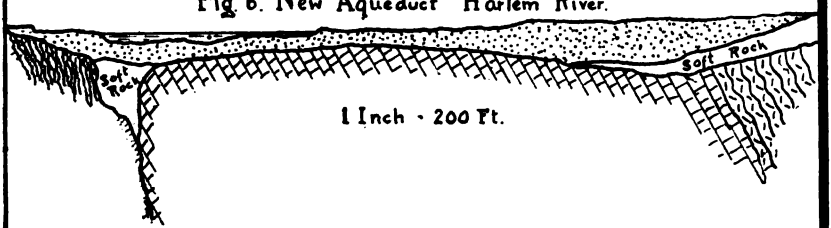


Fig 6. New Aqueduct Harlem River.



1 Inch = 200 Ft.

Fig 7. Rapid Transit Tunnel Harlem River.

1 Inch = 200 Ft.

Fig 8. Gas Tunnel, East River.

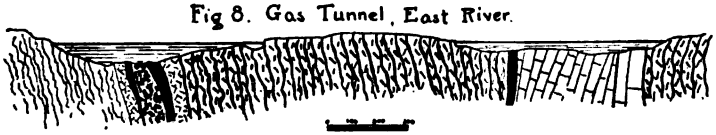
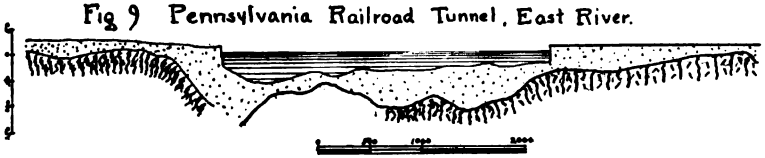


Fig 9 Pennsylvania Railroad Tunnel, East River.



Limestone Gneiss Schist

(Scale in Figs. 5, 6 and 7, as Reduced for Publication is 1 inch = 325 Feet.)

(Reports of other departments will appear in next issue.)

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with a depth of 200 ft. below the neighboring river bottom and a width of 1,200 ft., it appears by no means out of keeping with the rest of the profile. There is, of course, essentially no difference between the assumption of the presence of the gorge at Pegg's Point and that of its presence opposite New York—in neither case is it proved, but in each case it appears probable. At Cortlandt St. it can only be said that the channel must lie somewhere between the only two points at which rock was encountered—namely in a space of 3,000 ft.

Another point may profitably be noticed in the Pennsylvania section. In the space to which the channel has been assigned there is a slope on the west side of 1:6 and on the east of 1:3. In the map as given by the railroad these points are joined by a straight line, although it is very improbable that these slopes should thus cease abruptly. If they continued to a point they would give a depth of 434 ft.; and although they would hardly be expected to be so uniform, yet there is little doubt that the maximum depth is greater than that indicated on the map.

The evidence of the East and Harlem rivers may be taken for what it is worth; it may be said at least that in most places these rivers attained a remarkable depth, considering their widths and watersheds. In like manner the value of the evidence of the Hudsonian gorge is debatable. If it is the continuation of the Hudson River, there is without a doubt a double channel in the latter also; and conversely, if there is a buried channel in the Hudson it is very reasonable to consider the Hudsonian gorge, with its inner channel, as the continuation of the Hudson. A point of similarity between the two is the sinuosity of the inner channel with regard to the outer. This is noticed all along the course of the Hudsonian gorge and is actually shown in the Hudson at Cortlandt St., where the channel, howsoever deep it may be, swings over many hundreds of feet in the three miles between the Cortlandt and 32d St. crossings. The character of the gorge is not assured, however, and too great weight must not be attached to its evidence.

There remains one final and somewhat independent fact to be considered. At 110th St. and Manhattan Avenue, New York City, in the course of further explorations by the Water Board, a diamond drill hole has been sunk. After penetrating a compara-

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tively thin layer of glacial till much decomposed Manhattan schist was caught, which continued in this state to a depth of about 300 ft. The decomposition was evidently due to the circulation of underground waters—a lowering of the ground water level—although the rock was apparently not crushed to any great extent. Movement of water implies head, but there could have been no head unless these was an outlet at the bottom. Since this crushed zone is situated less than 3,000 ft. from the Hudson, it is reasonable to suppose that the water drained into that river, providing that a channel existed deep enough to supply the necessary head; otherwise there could have been no such extensive circulation. The decomposition of the rock at 300 ft. was so complete that it is probable that the waters percolated to a much greater depth, say at least 400 ft. Allowing 100 ft. more for head, to pull the water down into the interstices of the rock, we have a channel of at least 500 ft. indicated. This explanation seems to be the most simple, and it harmonizes moreover with the other facts cited above.

Conclusions.

It appears very probable, therefore, that there was originally an inner buried channel in the Hudson River some 200 ft. deeper at least than the outer one, and probably more, the existence of this channel as a separate entity being destroyed in some places (*e. g.*, Storm King) by subsequent glacial action. This channel is sinuous with regard to the outer one, and had originally fairly steep walls. It is probably to-day somewhat deepened by ice movement all along its course and locally to a greater extent, and in most places it is entirely filled with glacial material. Its presence indicates an immediately preglacial land elevation of 800–1,000 ft. over the present, for the channel is too sharply defined to admit of the supposition that base level had been reached when the ice put an end to further action. The river was probably still degrading strongly at that time. From the nature and amount of the data upon which these conclusions have been based, however, it is evident that a considerable element of speculation is involved; the only positive proof of the general existence of a buried inner channel would be the circumstantial evidence of drilling yet to be done.

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A FEW NOTES ON IRRIGATION.

By BORIS LEVITT, C.E.

In planning the construction of irrigation works it is a matter of vital importance to determine the quantity of water that can be made available for irrigation and that required to irrigate the area under consideration. To that end an investigation should be made of all conditions which directly or indirectly affect the availability and consumption of water.

In order to arrive at a correct estimate the climatic conditions prevailing at the locality must be carefully studied, as well as the nature of the soil to be irrigated, nature of crops grown, irrigation methods in use, etc. Improved methods of irrigation and intelligent treatment of the soil materially affect the quantity of water required, as all losses of water, whether in transit or through unskillful application to the land must be provided for or means taken to reduce them.

Duty.—The average duty of water in a canal is the ratio between the quantity of water which enters the canal and the number of acres it irrigates. This average duty is the one to be considered in planning irrigation works. It includes all losses incidental to the transportation of the water and the irrigation of the land.

The net duty of water is based upon the quantity of water supplied to the land as measured at the margin of the fields and does not include the losses which occur between the headgates of the canal and the margin of the fields. The net duty will be found to vary considerably in different localities. This variation in the duty is generally due to the extravagant and careless use of water, but frequently to unfavorable conditions for irrigation.

Units.—In hydraulic problems the unit of measurement is the cubic foot. In irrigation problems such a unit is too small, and the acre foot is used as the standard unit. There are other units employed in measuring water, but they are inaccurate and are dropping out of use.

An acre foot is the quantity of water that will cover an acre of

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land to a depth of one foot, and therefore contains 43,560 cubic feet. Since one foot of depth over the area of one acre represents an acre foot, the same quantity of water distributed over an area greater than one acre will give a lesser depth, which depth will numerically represent the fractional part of an acre foot per acre which the area contains. In other words, the numerical value of the depth to which a given volume of water would cover a given area will represent the number of acre feet per acre which that area contains.

Climate.—The necessity of irrigation and the duty of water are determined by the climate of the region. The rate of precipitation, the season during which it occurs, rate of evaporation, variations of temperature, frequency and intensity of winds, all exert an influence upon the quantity of water necessary to irrigate the land and the quantity available for such use. A proper investigation of climatic condition can be made only by continuous observations for a sufficiently long period of time.

Soil.—The duty of irrigating water is considerably affected by the nature of the soil to which it is applied. The amount of water the soil will retain, or the ease with which the water will percolate through it, will determine the quantity and frequency of irrigations necessary. A soil rich in clay will offer a greater resistance to percolation than a coarse gravelly one, and will retain what it has absorbed for a longer period. Frequency of irrigations necessary upon porous soils will result in a greater loss from evaporation and from percolation, and will result in a low duty.

Porous soils may be somewhat compacted by frequent irrigations. The water produces a tendency for the smaller particles of the soil to fill the pores between the larger ones and the ground gradually becomes closer, more compact and harder. A good growth of a deep-rooted vegetation will give the soil compactness and this is frequently resorted to by experienced cultivators.

Rainfall.—The necessity of irrigation is not always determined by the annual amount of rainfall, but by the amount of rainfall which is available during the growing season of the year.

In some parts of the arid region the average annual amount of rainfall would be sufficient for the production of crops provided it fell during the proper season of the year. This, however, is not

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generally the case, and in most arid regions the annual amount of rainfall, being distributed throughout the year, is not sufficient for the needs of the crops, the evaporation being considerably in excess of the rainfall.

Evaporation.—The rapidity of evaporation from water surfaces depends upon the relative temperature of the water and the atmosphere and the amount of motion in the latter. Evaporation is greatest when the air is driest, when the water is warm and when the velocity of the wind is high. It is least when humidity is highest, the air calm, and the temperature of the water low. The same is true of evaporation from soil surfaces. The evaporation from soil surfaces is also affected by the amount of moisture in the top layer of the soil. The evaporation is much more rapid from a saturated soil than from a soil containing a smaller quantity of water.

The evaporation from soil surfaces is much more rapid than from water surfaces under the same conditions, and a much greater loss of water by evaporation takes place after the water is spread over the field than while it is being conveyed in the canal.

During midsummer the soil becomes heated to a very high temperature, higher than the overlying air when exposed to the sun, and when a thin layer of water is spread over the heated land the evaporation is very rapid and the loss very high. The rate of evaporation rises rapidly with the temperature.

To check the rapid evaporation from the soil the fields are sometimes irrigated at night or in the early morning, when the rate of evaporation is not high. The loss by evaporation is also reduced by spreading the water over the field as quickly as possible, but this requires that the entire supply be available in a short period of time.

Losses of Water.—Of all the water that enters the headgates of an irrigation canal only a part of it reaches the irrigable lands. A considerable proportion, frequently as much as a half or more, is lost in transit in the main canal and laterals through evaporation, seepage and leakage. The duty of a canal, therefore, as measured by the quantity of water passing through the headgates of the main canal is much lower than if measured at a point where the water enters the irrigable lands.

The difference between the quantity of water that enters the canal

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and the quantity of water discharged by it on the land represents the total loss due to evaporation and seepage, which loss is entirely apart from the subsequent loss directly from the soil.

Although the total loss between any two points on the canal can be easily determined by direct measurement, the proportion of the loss caused by seepage is not directly determinable and can only be approximately obtained by estimating the loss due to evaporation. Leakage is due entirely to faulty construction and is preventable.

The serious losses of water are not entirely confined to the losses in transit. Of whatever amount of water reaches the fields after passing through the canals and laterals, a large proportion of the water is lost through evaporation and percolation directly from the soil.

Seepage.—Canals are subject to a large loss of water through percolation from the soil. The conditions affecting the loss in canals are the nature of the soil, character of the channel, velocity and volume of water conveyed. The nature of the soil, however, is the most important factor affecting the quantity of water lost. If the canal is built in clayey soil there will be little loss, but if the soil is either sandy or gravelly the loss will be very great. The loss is greatest in new canals and in a long line of canal may reach 50 per cent. or more of the volume entering it.

As the canal increases in age it becomes less pervious. The silt carried by the water in suspension is gradually deposited upon the banks and bottom, filling up the interstices and decreasing the amount of percolation. Some waters carry a cementing material which when deposited by the water on the banks and bottom make them almost impervious.

The absorption is greatly increased if the cross-section of the canal is bad. In this feature of construction an attempt should be made to reduce the wetted perimeter and the surface exposed to the water.

The rapid velocity of the water increases the porosity of the canal and the consequent loss by seepage, by removing the finer particles of the soil and leaving a bed of well-washed gravel which rapidly absorbs water.

The loss of water from seepage is also affected by the volume of

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water carried by the canal. If the water is held at the same level for a long period of time there will be little change in the volume lost by seepage. As soon, however, as the water in the canal is raised, the pressure is increased and new surfaces uncoated by silt come in contact with the water and increase the loss.

Various methods have been tried to reduce excessive seepage in canals. To puddle the canal with clay, to line with tarred cloth, to throw fine clay into the water and let the water carry it down and deposit it on the bottom and sides of the canal, to maintain the water in the canal at the same level for a long time. A ditch lined with tarred cloth will carry the water almost without loss, with loose clay the loss is reduced so long as the surface of the canal is not disturbed, but the application of the clay has to be renewed each season.

It frequently happens that canals, instead of losing, gain water. This is due to the fact that more water seeps into the canal from surrounding high country than out of it.

The seepage losses continue after the water is distributed on the field. The percolating water from canals and irrigated fields frequently increase the flow of adjacent streams. Measurements of the return water have shown that it reaches in many instances 30 per cent. of their original volume.

Ground Water.—Continued irrigation on new land tends to raise the level of the ground water nearer the surface. New land will continue to absorb a large amount of irrigation water and store it in the form of ground water until the subsoil becomes saturated. Deep-rooted plants derive a great deal of their nourishment from this source and wherever the ground water is within reach of the plants the duty for the land becomes correspondingly higher. The duty for new land rises after a few seasons of irrigation.

In some parts of the irrigated region the farmers make use of the plentiful supply of water in the winter by flooding their fields during the winter with the intention of storing the water in the land in anticipation of a probable shortage of water during the growing season.

Irrigation and Service Periods.—Irrigation period is the period of time during one season, determined by the time between the first and last watering of the season.

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Service period is the time during which the water is allowed to flow during one watering.

The irrigation period varies considerably for different sections, depending upon the duration of the growing season. For some canals the irrigation season lasts all year, for some only perhaps two months. The duration of the irrigation period with respect to certain crops is uncertain, scientific cultivation not having yet determined it.

No data are yet available to establish a direct relation between the duration of the irrigation period and the duty of water, so far as the application of water is concerned. The duty is probably lowered through evaporation from the soil during long irrigation periods.

The duration of the service periods will largely depend upon the total quantity of water required by any particular crop during the irrigation period and the capacity of the irrigating ditches. The nature of the crop, too, has much to do with it. These things, however, are controlled by considerations peculiar to individual irrigators and local experience.

Method of Irrigation.—There are two general methods of irrigation in use: by flooding and by furrows.

In the first method a system, generally known as the check system, is sometimes used, in which the land is built up into a number of checks or sections of level ground surrounded by levees to confine the water. The checks are watered successively by water passing from one check to another, or they may be all watered at the same time by laterals which conduct the water along their borders.

The furrow system consists of running furrows through the land to be irrigated in rows about three feet apart. The upper ends of these furrows connect to the head ditch or flume, from which the water is turned into furrows and allowed to follow down their course until the ground is thoroughly wet. The amount of water to be turned into any furrow depends upon the slope and capacity of the soil to absorb it. In using the furrow system there is a saving in the water lost from the soil. The saving is greater for deeper furrows.

The loss of water from the soil is greater when the water is

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applied in frequent light irrigations. This practice keeps the water always exposed to conditions causing the largest evaporation. Unless the nature of the soil or of the crops demands such irrigations much water can be saved by irrigating less often.

Excessive Irrigation.—Excessive use of water in irrigation is one of the causes of low duty of water for the land. Excessive watering is generally of little benefit to the crops and does much harm to the land.

Where the subsoil is porous and permits of free percolation the water in escaping into the subsoil below removes many of the soluble ingredients by leaching and carries them away into the low lying lands to appear there as harmful alkali, destructive to vegetation. The effect of removing these salts is that in removing the hurtful alkali the useful salts are removed too, in this way impoverishing the land and converting the low lying lands into swamps. Many acres of such low lying lands have been converted into swamps through irrigation waters and became charged so heavily with alkali that they would produce nothing but coarse aquatic plants.

Application of Water.—Application of water at the proper time during the growth of the crops will produce a marked influence upon the total amount of water that will be needed to bring the crops to maturity. In the case of some crops the application of too much water in their early growth will result in the formation of surface roots, which will require frequent irrigations to keep them supplied with the necessary moisture. On the other hand the application of water during the later stages of their growth will require a smaller quantity of water for their development.

The proper time for the application of water generally varies with the individual irrigators even for the same crops in the same section of the country. A correct knowledge of the proper time for the application of water would be an important factor in the determination of the duty of water for any particular section.

The season of the year during which crops may be raised has some effect upon the duty of water. In some sections of the West where winters are not severe an early planting or seeding of some crops saves much of the irrigated water from the loss by evaporation.

It was observed in the case of some crops that if planted soon

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after the coolest weather is gone, a greater yield is secured with a smaller quantity of water than if planted later in the season. These crops are planted early and make most of their growth before the season of intense evaporation arrives. The possibility of raising crops during such an early season of the year will save a great deal of the water otherwise lost by evaporation.

Cultivation.—Thorough cultivation of the soil following an irrigation reduces the loss of water from evaporation and consequently the frequency of irrigations. Cultivation of the soil after an irrigation breaks up the openings through which the moisture finds its way out of the soil; it also forms a mulch on the surface which prevents rapid evaporation.

Distribution.—The form of contracts for the supply of irrigation water, in general use, provides for the delivery of a continuous flow and the irrigator is charged without regard to the actual quantity consumed by him. With such an arrangement there is no inducement to the irrigator to use his supply economically, and the result is a wasteful use of the water and a correspondingly low duty.

The safest way to increase the duty of water is to promote economy in its use. As long as the irrigator has to pay for the water, regardless of the quantity he uses, he will not make any attempt to economize, but should he be given an opportunity to reduce his water bill he will surely make an attempt to do so.

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A LIST OF NEW CRYSTAL FORMS OF MINERALS.—II.*

By H. P. WHITLOCK.

Edingtonite (Dana's orientation).

O. Nordenskjold. Geol. Fören. Förh., 1895, *7*, 597.
Base: $c(001)$.
Sphenoid: $p_1(111)$.
Scalenohedrons: $r(121)$, $r_1(121)$.
H. Sjögren. Geol. Fören. Förh., 1906, *28*, 169.
Scalenohedrons: $(33\bar{2})$, $(1.10.\bar{9})$.

Eglestonite.

Isometric.

A. J. Moses. Am. Jour. Sci., 1903, *16*, 253.
Cube: $a(110)$.
Dodecahedron: $d(110)$.
Trapezohedron: $n(112)$.
Hexoctahedron: $s(123)$.
W. T. Schaller. U. S. Geol. Surv., Bull. 405, 1909, 147.
Octahedron: $o(111)$.
Tetrahexahedrons: $e(120)$, $f(130)$.
Trapezohedron: $\phi(116)$.
Trisoctahedrons: $r(332)$, $p(221)$, $\rho(441)$.
Hexoctahedrons: $i(189)$, $j(167)$, $w(156)$, $k(146)$, $\Sigma(145)$, $F(126)$, $v(4.7.11)$, $M(234)$, $l(347)$.

Elpidite.

Orthorhombic.

G. Lindström and G. Nordenskiöld. Geol. Fören. Förh., 1894, *16*, 330 and 343.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $u(540)$?, $m(110)$, $t(580)$?, $n(120)$, $s(5.12.0)$?.

* Continued from Vol. XXXI., July 1910.

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Domes: $e(013)$, $d(011)$.

G. Flink. Medd. om Grönland, 1898, *14*, 230.
Dome: (102).

ENARGITE.

L. V. Pirsson. Am. Jour. Sci., 1894, *47*, 212.
Prism: $v(210)$.
Pyramid: $z(134)$.

L. J. Spencer. Min. Mag., 1897, *11*, 69.
Prisms: $y(610)$, $f(520)$, $i(540)$, $N(230)$.
Domes: $t(108)$, $A(207)$, $w(709)$.
Pyramids: $\phi(132)$, $\phi_2(131)$, $\phi_1(394)$, $\phi_3(392)$.

S. Stevanovič. Zeitschr. f. Kryst., 1902, *37*, 24.
Prisms: $s(410)$, $T(430)$, $F(980)$, $P(250)$, $L(270)$, $R(140)$, $D(160)$.
Domes: $B(205)$, $G(041)$.

A. J. Moses. Am. Jour. Sci., 1905, *20*, 277.
Prisms: $h(120)$, $l(130)$.
Dome: $\theta(051)$.
Pyramid: $P(223)$.

ENDEIOLITE.

Isometric.

G. Flink. Medd. om Grönland, 1901, *24*, 166.
Octahedron: $o(111)$.

EPIDIDYMITE.

Orthorhombic.

G. Flink. Geol. Fören. Förh., 1893, *15*, 201.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $m(110)$, $l(120)$, $n(130)$.
Domes: $i(023)$, $h(034)$, $g(011)$, $e(043)$, $d(021)$, $f(041)$, $\lambda(061)$, $\chi(081)$.
Pyramid: $p(221)$.

G. Flink. Geol. Fören. Förh., 1899.
Prism: (310).
Dome: (301).
Pyramid: (312).

G. Flink. Bull. geol. Inst. Upsala, 1898, *4*.
Domes: (015), (058).

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EPIDOTE.

E. Artini. Rend. Acc. Lincei, 1887, *4*, 380.
Prism: (350).
Domes: (10.0.1), (905), (13.0.3), (19.0.14), (504), (13.0.12), (22.0.23).
Pyramids: (994), (552), (331), (767), (8.15.8), (3.17.3), (1.10.1), (1.13.1), (11.20.20), (9.14.14), (15.16.16), *X*(322), (21.10.10), (833), (23.10.7), (28.12.9), (26.11.8).

L. Brugnatelli. Zeitschr. f. Kryst., 1890, *17*, 259.
Dome: $\mathfrak{v}$(014).

J. Gränzer. Min. Mitth., 1888, *9*, 361.
Pyramids: (511), (711), (811).

F. Zambonini. Zeitschr. f. Kryst., 1901, *34*, 225, and 1903, *37*, 1.
Domes: (39.0.1), (13.0.11), (203).
Pyramids: (14.14.1), (744).

F. Zambonini. Rend. Acc. Lincei, 1903, *12*, 567.
Domes: (801), (501).

C. Palache. Proc. Am. Ass. Sci. Arts, 1902, *37*, 531.
Pyramids: *O*(544), *j*(755).

Z. Toborffy. Zeitschr. f. Kryst., 1907, *43*, 564.
Domes: (13.0.1), (12.0.1), (11.0.5), (19.0.50), (7.0.20), (5.0.11), (23.0.50), (503), (25.0.12), (11.0.6), (703), (502), (801), (901), (11.0.1).
Pyramids: $\mathfrak{m}$(117), $\mathfrak{z}$(118), $\mathfrak{n}$(119), (1.1.19), (1.1.25), *Y*(214).

G. Flink. Bihang t. K. Sv. Vet. Akad. H. Stockholm, 1886, *12*, No. 2.
Pyramid: *F*(454).

EPISTOLITE.

Monoclinic.

O. B. Böggild. Medd. om Grönland, 1900, *24*, 183.
Pinacoid: *c*(001).
Prism: *m*(110).
Domes: *s*(102), *r*(504), *o*(011).

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ERIKITE.

Orthorhombic.

O. B. Böggild. Medd. om Grönland, 1903, *26*, 93.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $m(110)$, $n(120)$, $o(130)$, $p(270)$.
Domes: $r(101)$, $s(201)$, $d(012)$, $e(011)$, $f(032)$, $g(021)$, $h(052)$, $i(031)$.
Pyramids: $u(114)$, $t(111)$.

EUDIDYMITE.

G. Flink. Medd. om Grönland, 1898, *14*, 230.
Dome: (053).
Pyramids: (15.5.3), (15.5.3), (625).
G. Flink. Bull. geol. Inst. Upsala, 1898, *4*.
Pyramids: (221), ($\bar{1}14$), (338), ($\bar{5}59$).

FICHTELITE.

F. Plzák and V. Rosicky. Zeitschr. f. Kryst., 1908, *44*, 332.
Domes: $e(\bar{2}01)$, $x(\bar{3}04)$, $z(0\bar{1}1)$.
Pyramid: $y(11\bar{4})$.
H. Böckh. Földt. Közlöny, 1904, *34*, 335.
Prism: $o(\bar{1}10)$.

FLORENCITE.

Hexagonal-rhombohedral.

E. Hussak and G. T. Prior. Min. Mag., 1900, *12*, 244.
Base: $c(0001)$.
Prism: $m(10\bar{1}0)$.
Rhombohedrons: $r(10\bar{1}1)$, $f(02\bar{2}1)$.

FLUORITE.

M. Weber. Zeitschr. f. Kryst., 1903, *37*, 433.
Trapezohedrons: (17.1.1), (10.1.1), (8.7.7).
Hexoctahedrons: (931), (19.9.5).
H. P. Whitlock. N. Y. State Mus., Bull. 140, 1910, 198.
Trapezohedron: ⊛(19.1.1).
A. Krejči. Abh. d. bohm. Ak., 1905, No. 2, 3.
Hexoctahedron: $T(20.4.3)$.

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Hulyák. Földt. Közlöny, 1903, *33*, 54 and 175.
Hexoctahedrons: $j(15.7.4)$, $r(24.10.7)$.

GALENA.

A. F. Rogers. Am. Jour. Sci., 1901, *12*, 45.
Trapezohedron: $\rho(13.1.1)$.
G. Cesàro. Zeitschr. f. Kryst., 1892, *20*, 468.
Trisoctahedron: (551).
L. Colomba. Rend. Acc. Lincei, 1905, *15*, 636.
Trisoctahedron: (661).

GARNET.

A. Catherine. Min. Mitth., 1888, *10*, 55 and 397.
Hexoctahedrons: $D(3.7.10)$, $F(358)$.
C. Palache and H. O. Wood. Am. Jour. Sci., 1904, *18*, 344.
Hexoctahedron: (459).
F. Zambonini. Zeitschr. f. Kryst., 1904, *40*, 225.
Hexoctahedron: (235).

GEIKIELITE.

Hexagonal.

P. P. Sustschinsky. Zeitschr. f. Kryst., 1902, *37*, 57.
Base: $c(0001)$.
Rhombohedron: $\phi(5058)$.

GEOCRONITE.

Orthorhombic (new orientation).

G. d'Achiardi. Att. Soc. Tosc. Mem., 1901, *18*.
Pinacoid: $c(001)$.
Prism: $m(110)$.
Dome: (058), (067), (011), (032), (021).
Pyramids: (225), (111).

GEORGIADÉSITE.

Orthorhombic.

A. Lacroix and A. de Schulten. Compt. rend., 1907, *145*, 783.
Pinacoids: (100), (010).
Domes: (011), (0.11.4).
Pyramids: (451), (16.5.4), (4.15.2).

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GOLD.

P. Jeremejeff. Vh. russ. Min. Ges., 1895, *33*.
Trapezohedron: (811).

GRAFTONITE.

Monoclinic.

S. L. Penfield. Am. Jour. Sci., 1900, *9*, 20.
Pinacoids: $a(100)$, $b(010)$.
Prisms: $m(110)$, $l(120)$, $n(130)$.
Domes: $d(011)$, $e(021)$.
Pyramid: $p(111)$.

GYPSUM.

O. Luedecke. Min. d. Harzes, 1896, 377.
Prisms: $\phi(350)$, $p(510)$.
A. Lacroix. Bull. soc. min., 1898, *21*, 39.
Dome: $(\bar{2}03)$.
Pyramids: (211), $(\bar{5}49)$, $(15.21.\bar{2}6)$.
A. F. Rogers. School of Mines Quar., 1902, *23*, 133.
Prism: (13.7.0).
A. Artini. Rend. R. Inst. Lomb., 1903, *36*, 11 and 81.
Pyramids: $o(\bar{2}12)$, $p(313)$.
G. Cesàro. Bull. Ac. Belg., 1905, 140.
Pyramid: $(\bar{3}14)$.

GYROLITE.

Hexagonal-rhombohedral.

O. B. Böggild. Medd. om Grönland, 1908, *34*, 93.
Base: $c(0001)$.
Rhombohedrons: $r(10\bar{1}1)$, $u(10\bar{1}2)$.

HACKMANITE.

Isometric.

L. H. Borgström. Geol. Fören. Förh., 1901, *23*, 563.
Dodecahedron: $d(110)$.

HAMLINITE.

Hexagonal-rhombohedral.

S. L. Penfield. Am. Jour. Sci., 1897, *4*, 313.
Rhombohedrons: $r(10\bar{1}1)$, $f(02\bar{2}1)$.

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HANCOCKITE.

Monoclinic.

S. L. Penfield and C. H. Warren. Am. Jour. Sci., 1899, *8*, 339.
Pinacoids: $a(100)$, $c(001)$.
Domes: $e(101)$, $r(\bar{1}01)$.
Pyramid: $n(\bar{1}11)$.

HAUCHECORNITE.

Tetragonal.

R. Scheibe. Zeitschr. Geol. Ges., 1888, *40*, 611.
Base: $c(001)$.
Prisms: $a(100)$, $m(110)$.
Pyramids: $e(101)$, $s(112)$, $p(111)$.

HAUERITE.

V. Goldschmidt and Schröder. Zeitschr. f. Kryst., 1908, *45*, 214.
Trisoctahedron: $\beta(477)$.

HAUSMANNITE.

G. Flink. Bihang. t. K. Sven. Vet. Ak., *16* (2), 4.
Pyramids: (414), (223), (335), (5.5.11).

HAUTEFEUILLITE.

Monoclinic.

Michel. Bull. soc. fr. Min., 1893, *16*, 38.
Pinacoids: $a(100)$, $b(010)$.
Prism: $m(110)$.

HELLANDITE.

Monoclinic.

W. C. Brögger. Zeitschr. f. Kryst., 1906, *42*, 417.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $m(110)$, $l(120)$, $g(540)$, $n(320)$.
Domes: $x(101)$, $d(\bar{1}02)$, $r(\bar{1}03)$, $t(205)$, $e(201)$, $g(301)$, $o(011)$.
Pyramid: $p(\bar{1}22)$.

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HEMATITE.

F. Gonnard. Compt. rend., 1898, *126*, 1048.
Rhombohedron: (1015).
Scalenohedrons: (4265), (25.24.49.47), (13.12.25.19), (27.24.51.39), (17.10.27.22), (15.8.23.19), (3.10.13.8), (35.48.83.59).

Dufet. Bull. soc. fr. Min., 1903, *26*, 60.
Scalenohedron: (1344).

R. W. Harre. Zeitschr. f. Kryst., 1906, *42*, 280.
Scalenohedrons: (10.5.15.12), (2465), (2133).

HERDERITE.

Monoclinic.

S. L. Penfield. Am. Jour. Sci., 1894, *47*, 329.
Pinacoids: *a*(100), *b*(010), *c*(001).
Prisms: *m*(110), *l*(120), *μ*(130).
Domes: *d*(101), *e*(302), *ɛ*(302), *b*(301), *u*(011), *t*(032), *v*(031), *s*(061).
Pyramids: *r*(112), *p*(111), *q*(332), *n*(331), *o*(441), *q*(332), *n*(331), *k*(112), *w*(3.12.4), *r*(121), *x*(362), *z*(394), *p*(391).

HEULANDITE.

Goodchild. Trans. Geol. Soc. Glasgow, 1903, *12*, Sup. 1.
Dome: *w*(032).
Pyramid: *n*(111).

O. B. Böggild. Medd. om Grönland, 1905, *28*, 120.
Dome: (501).

HIBSCHITE.

Isometric.

F. Cornu. Min. Mitth., 1905, *24*, 327, and 1906, *25*, 249.
Dodecahedron: *d*(101).
Octahedron: *o*(111).

HOPEITE.

H. Buttgenbach. Ann. Soc. Géol. belg., 1906, *33*, M. 9.
Prism: (130).

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HUTCHINSONITE.

Orthorhombic.

R. H. Solly. Min. Mag., 1905, *14*, 72.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: (850), (870), $m(110)$, (780), $h(340)$, (580), $f(120)$, (380), $g(140)$, (180).
Domes: (502), $u(201)$, $W(302)$, $d(101)$, (304), $U(102)$, (104), (011).
Pyramids: (322), $p(111)$, (344), (122), (144).

G. F. H. Smith and G. T. Prior. Min. Mag., 1907, *14*, 284.
Prisms: $F(210)$, $l(740)$, $k(320)$, $i(540)$.
Domes: $v(301)$, $w(401)$.
Pyramids: $N(221)$, $n(121)$, $t(342)$, $q(322)$, $o(211)$, $Q(311)$.

HUMITE GROUP.

CHONDRODITE.

H. Sjögren. Geol. Fören. Förh., 1892, *14*, 423.
Pyramids: (115), $(\bar{1}12)$, (115), (327), (3.2.11), $(\bar{2}12)$.

ILMENITE.

L. Piolti. Atti. d. R. Ac. Sci. Turin, 1902, *36*, = Mohsite.
Second Order Rhombohedron: (1122).
First Order Rhombohedrons: (0.4.4.11), (2027).

P. P. Sustschinsky. Zeitschr. f. Kryst., 1902, *37*, 60.
Third Order Rhombohedron: $y(3142)$.

R. H. Solly. Min. Mag., 1905, *14*, 184.
Prism: $h(4\bar{1}50)$.
Second Order Rhombohedrons: $\chi(44\dot{8}3)$, $\beta(5.5.10.24)$.
First Order Rhombohedrons: $\alpha(0.7.7.20)$, $r((40\dot{4}1)$, $d(10\dot{1}2)$.
Third Order Rhombohedrons: $\gamma(12\bar{3}2)$, $\delta(24\dot{6}7)$, $q(6\dot{2}4\dot{1})$, $\kappa(3\dot{1}2\dot{1})$.

ILVAITE.

O. B. Böggild. Medd. om Grönland, 1902, *25*, 45.
Prism: $\gamma(160)$.
Domes: $\sigma(081)$, $f(041)$, $i(401)$.
Pyramids: $m(621)$, $\zeta(331)$, $\epsilon(431)$.

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Inesite.

A. Hamberg. Geol. Fören. Förh., 1894, *16*, 323.
Dome: $f(301)$.

O. C. Farrington. Field Colum. Mus. Pub., 1900, *1* (7), 221.
Dome: $k(11.0.12)$.
Pyramid: $s(\bar{9}46)$.

Iodyrite.

E. H. Kraus and C. W. Cook. Am. Jour. Sci.. 1909, *27*, 210.
Rhombohedrons: $r(7074)$, $t(7073)$, $x(7071)$, $y(9091)$, $y'(909\bar{1})$, $w(9092)$, $w'(\bar{9}092)$, $s'(15.0.15.8)$, $z(3032)$.

Iolite.

H. Gemböck. Zeitschr. f. Kryst., 1898, *29*, 305.
Prisms: (350), (120), (160).
Dome: (501).
Pyramids: (351), (261), (281), (212), (4.12.3), (3.18.1), (164), (168).

Jordanite.

(Shown to be Monoclinic by Baumhauer.)

H. Baumhauer. Sitz. Ber. Ak., 1891, 697 and 915.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $8r(180)$, $11\frac{1}{2}r(2.11.0)$, $15\frac{1}{2}r(2.15.0)$, $5r(150)$, $9\frac{1}{2}r(290)$, $49/12(12.49.0)$, $4r(140)$, $32/9r(9.32.0)$, $7\frac{1}{2}r(270)$, $24/7r(7.24.0)$, $3r(130)$, $11\frac{1}{4}r(4.11.0)$, $5\frac{1}{2}r(250)$, $7/3r(370)$, $27/12r(12.27.0)$, $2r(120)$, $3\frac{1}{2}r(230)$, $5\frac{1}{4}r(450)$, $7/6r(670)$, $r(110)$, $2s(210)$, $4s(410)$.
Domes: $-h(101)$, $+h(\bar{1}01)$, $-2h(201)$, $+2h(\bar{2}01)$, $-3h(301)$, $+3h(\bar{3}01)$, $-5h(501)$, $+5h(\bar{5}01)$, $\frac{1}{2}k(072)$, $3k(031)$, $\frac{1}{2}k(052)$, $2k(021)$, $\frac{3}{2}k(032)$, $k(011)$, $\frac{1}{2}k(012)$.
Pyramids: $+18q(1.18.1)$, $-17q(1.17.1)$, $+12q(\bar{1}.12.1)$, $-10q(1.10.1)$, $+10q(\bar{1}.10.1)$, $-9q(191)$, $+9q(\bar{1}91)$, $-8q(181)$, $+8q(\bar{1}81)$, $-7q(171)$, $+7q(\bar{1}71)$, $-6q(161)$, $+6q(\bar{1}61)$, $-5q(151)$, $+5q(\bar{1}51)$, $-4q(141)$, $+4q(\bar{1}41)$, $-\frac{1}{2}q(272)$, $+\frac{1}{2}q(\bar{2}72)$, $-3q(131)$, $+3q(\bar{1}31)$, $-2q(121)$, $+2q(\bar{1}21)$, $-p(111)$, $+p(\bar{1}11)$, $-2t(212)$, $+2t(\bar{2}12)$, $+28/3t(\bar{2}8.3.28)$, $-6x(361)$, $+6x(\bar{3}61)$, $-5x(351)$, $+5x(\bar{3}51)$, $-7x(371)$, $-8x(381)$, $-4x(341)$, $+4x(\bar{3}41)$,

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$-3p(331)$, $+3p(\bar{3}31)$, $-5v(521)$, $-3v(321)$, $+3v(\bar{3}21)$, $-3u(311)$, $+3u(\bar{3}11)$, $+3z(\bar{2}31)$, $+\frac{5}{2}z(452)$, $-2u(211)$, $+2u(\bar{2}11)$, $-2y(412)$, $+2y(\bar{4}12)$.

R. H. Solly. Zeitschr. f. Kryst., 1901, *35*, 321.

Prisms: (340), (310), (510).

Pyramids: ($\bar{1}$.17.1), ($\bar{1}$.14.1), ($\bar{1}$.13.1), (2.11.2), (292), ($2\bar{5}2$), (232), (432).

KAINITE.

C. Busz. Ber. Med.-naturwiss. Ges., Münster, 1906.

Prisms: $u(510)$, $f(340)$, $g(230)$, $h(130)$.

Dome: $i(021)$.

Pyramid: $w'(\bar{3}11)$.

KENTROLITE.

G. Nordenskjold. Geol. Fören. Förh., 1894, *16*, 153.

Dome: $e(102)$.

Pyramids: $u(114)$, $v(115)$, $z(3.15.10)$.

KLEINITE.

Hexagonal.

A. Sachs. Sitzungsber. K. preuss. Akad. Wiss. Berlin, 1905, 1091.

Base: $c(0001)$.

Prisms: $m(10\bar{1}0)$, $a(11\bar{2}0)$.

Pyramids: $p(10\bar{1}1)$, $x(10\bar{1}2)$.

KRENNERITE.

G. F. Herbert Smith. Min. Mag., 1903, *13*, 264.

Prisms: $K(310)$, $L(520)$, $M(650)$, $N(340)$, $j(140)$, $J(160)$.

Domes: $\delta(043)$, $\theta(054)$, $\mu(104)$, $\Pi(502)$, $\pi(401)$, $\lambda(901)$.

Pyramids: $\beta(141)$, $\gamma(581)$, $f(231)$, $\alpha(562)$, $r(142)$, $x(342)$, $R(542)$, $\phi(343)$, $\chi(144)$, $\psi(344)$, $\xi(411)$.

KRÖHNKITE.

Monoclinic (new orientation).

C. Palache and C. H. Warren. Am. Jour. Sci., 1908, *26*, 342.

Pinacoids: $a(100)$, $b(010)$.

Prisms: $h(120)$, $k(130)$.

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Domes: $e(011)$, $d(021)$, $f(031)$, $t(\bar{1}01)$, $u(\bar{3}02)$, $v(\bar{3}01)$.
Pyramids: $p(111)$, $r(121)$, $q(\bar{1}11)$, $s(\bar{1}21)$, $w(\bar{2}11)$, $x(\bar{2}21)$, $z(\bar{3}31)$, $i(551)$, $o(\bar{1}0.10.1)$, $y(\bar{2}32)$, $n(132)$.

Laurionite.

G. F. Herbert Smith. Min. Mag., 1900, *12*, 102.
Pyramids: $o(112)$, $q(122)$, $r(132)$, $s(142)$, $t(152)$, $u(232)$.

Leadhillite.

E. Artini. Gior. d. Min., 1890, *1*, 1.
Prism: (430).
Domes: (401), (302), ($\bar{1}$01).
Pyramids: (113), (434), (418), ($\bar{4}$18), ($\bar{4}$.1.12), (477), (214), (612), ($\bar{6}$36), (436).

Leonite.

Monoclinic.

C. A. Tenne. Zeitschr. Geol. Ges., 1896, *48*, 632.
Pinacoids: $b(010)$, $c(001)$.
Prism: $\mu(120)$.
Domes: $d(102)$, $\delta(\bar{1}02)$, $o(013)$, $n(011)$.
Pyramids: $q(113)$, $p(111)$, $\pi(\bar{1}11)$.
J. E. Strandmark. Zeitschr. f. Kryst., 1902, *36*, 461.
Dome: $w(101)$.
Pyramid: $(\bar{2}11)$.

Lepidolite.

W. T. Schaller. Am. Jour. Sci., 1905, *19*, 225.
Prism: $l(130)$.
Pyramids: $o(112)$, $u(111)$, $x(131)$, $N(261)$, $z(\bar{1}32)$, (223)?, (221)?, (112)?.

Leucosphenite.

Monoclinic.

G. Flink. Medd. om Grönland, 1898, *14*, 236, and 1901, *24*, 137.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $m(110)$, $n(130)$.
Domes: $d(101)$, $x(011)$.
Pyramids: $s(112)$, $p(\bar{1}11)$, $r(263)$, $g(133)$.

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LINARITE.

G. Cesàro. Bull. acad. R. Belg., 1905, 328.
Pyramid: $i(12.2.11)$.

O. C. Farrington and E. W. Tillotson. Field Col. Mus. (Geol.), 1908, *3*, 148.
Domes: $\delta(10.0.9)$, $\phi(\bar{9}.0.10)$.
Pyramid: $f(523)$.

LORANDITE.

Monoclinic.

V. Goldschmidt. Zeitschr. f. Kryst., 1898, *30*, *272*.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $q(210)$, $\epsilon(320)$, $m(110)$, $e(120)$, $\mu(130)$, $u(140)$?.
Domes: $\beta(\bar{2}05)$, $d(101)$, $\kappa(201)$, $\alpha(034)$, $h(045)$, $w(021)$.
Pyramids: $v(112)$, $s(\bar{1}12)$, $z(\bar{1}\bar{1}1)$, $g(425)$, $f(212)$, $\gamma(748)$, $j(536)$, $i(324)$?, $l(122)$, $\delta(1.13.8)$?, $y(312)$, $n(525)$, $p(\bar{2}12)$, $r(\bar{2}11)$, $\zeta(\bar{3}24)$, $k(322)$, $\eta(\bar{2}.5.10)$?.

LORENZENITE.

Orthorhombic.

G. Flink. Medd. om Grönland, 1898, *14*, 250, and 1901, *24*, 130.
Pinacoids: $a(100)$, $b(010)$.
Prisms: $m(110)$, $n(120)$, $x(1.12.0)$.
Pyramids: $p(111)$, $o(231)$.

MAGNETITE.

H. P. Whitlock. N. Y. State Mus., Bull. 140, 1910, 199.
Trapezohedron: $\Psi(711)$.

MALACHITE.

G. Cesàro. Bull. Ac. Belg., 1904, 1206.
Pyramid: $(\bar{4}23)$.

MANGANITE.

F. Zambonini. Zeitschr. f. Kryst., 1901, *34*, *225*.
Prisms: $E(320)$, $L(940)$, $R(560)$, $U(590)$.

R. Busz. Zeitschr. f. Kryst., 1889, *15*, 616.
Pyramid: (121).

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MARRITE.

Monoclinic.

R. H. Solly. Min. Mag., 1905, *14*, 76, and 1906, *14*, 184.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: (720), (310), (210), (320), (110), (230), (120), (130), (140), (150), (170).
Domes: (201), (013), (012), (023), (011), (021), (073), $\frac{8}{3}k(083)$, (072), $4k(041)$.
Pyramids: (111), (112), (223), ($\bar{1}$11), $-v(312)$, (212), (211), $-3q(131)$, (121), $-5q(151)$, $-w(271)$, (211), (233), (121).

MELANOTEKITE.

Orthorhombic.

G. Nordenskiöld. Geol. Fören. Förh., 1894, *16*, 158.
Pinacoid: $a(100)$.
Prism: $m(110)$.
Dome: $d(101)$.
Pyramids: $o(111)$, $s(221)$.
C. H. Warren. Am. Jour. Sci., 1898, *6*, 116.
Pinacoid: $b(010)$.
Prisms: $n(130)$, $k(150)$.

MELANTERITE.

W. T. Schaller. Bull. Univ. Calif. (Geol.), 1903, *3*, 195.
Prism: $l(120)$.
Domes: $d(102)$, $k(203)$, $x(302)$, $q(201)$, $j(904)$.
Pyramid: $B(332)$.

MERCURIC IODIDE.

Isometric.

A. J. Moses. Am. Jour. Sci., 1901, *12*, 98.
Cube: (100).

MIARGYRITE.

A. S. Eakle. Zeitschr. f. Kryst., 1899, *31*, 209.
Domes: $T(106)$, $V(1.0.12)$, $U(1.0.30)$, $W(1.0.12)$.
Pyramids: $Z(913)$, $K(112)$, $H(411)$, $P(15.2.2)$.

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MILLERITE.

C. Palache and H. O. Wood. Am. Jour. Sci., 1904, *18*, 343.

Prisms: $d(7290)$.

(Uncertain) $f(9.4.13.0)$, $g(31.13.\bar{44}.0)$.

Rhombohedrons: $p(0221)$.

(Uncertain) $x(40\bar{4}1)$, $j(5051)$, $h(3031)$, $l(0991)$, $m(0.18.\bar{18}.1)$.

Scalenohedrons: $s(21\bar{3}1)$, $u(1453)$.

(Uncertain) $o(7.4.\bar{11}.9)$, $q(6174)$, $n(5276)$, $w(42\bar{6}5)$.

MOLYBDENITE.

Hexagonal.

A. P. Brown. Proc. Acad. Nat. Sci. Phila., 1896, 210.

Base: $c(0001)$.

Prism: $m(10\bar{1}0)$.

Pyramids: $p(20\bar{2}1)$, $o(1011)$, $q(3031)$.

A. J. Moses. Am. Jour. Sci., 1904, *17*, 365.

Pyramids: $r(50\bar{5}4)$, $t(2023)$, $s(20\bar{2}5)$, $u(1014)$.

MONAZITE.

H. S. Bowman. Min. Mag., 1900, *12*, 349.

Prism: $\mu(130)$.

Pyramids: $\eta(\bar{1}32)$, $\theta(\bar{1}22)$.

MONTROYDITE.

Orthorhombic.

A. J. Moses. Am. Jour. Sci., 1903, *16*, 253.

Pinacoids: $a(100)$, $b(010)$.

Prism: $m(110)$.

Dome: $d(101)$.

Pyramids: $s(112)$, $o(111)$, $x(331)$, $e(132)$, $t(122)$, $r(211)$, $w(311)$.

W. F. Hillebrand and W. T. Schaller. U. S. Geol. Surv., Bull. 405, 1909, 47.

Prisms: $l(1.10.0)$, $h(120)$, $k(350)$, $C(230)$, $f(320)$, $\xi(310)$, $j(410)$, $\Phi(510)$.

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Domes: $v(012)$, $y(023)$, $K(045)$, $z(011)$, $G(032)$, $L(021)$, $\beta(061)$, $E(103)$, $g(102)$, $M(203)$, $n(302)$, $q(201)$, $\mu(301)$.
Pyramids: $\alpha(133)$, $\zeta(265)$, $N(263)$, $Z(376)$, $O(243)$, $Q(239)$, $S(232)$, $R(346)$, $U(344)$, $A(114)$, $B(113)$, $D(223)$, $i(221)$, $\delta(551)$, $V(5.4.10)$, $\gamma(542)$, $P(326)$, $\pi(323)$, $\phi(632)$, $\rho(7.3.13)$, $\lambda(317)$, $\Delta(313)$, $W(411)$, $\omega(10.1.12)$.

MUSCOVITE.

H. Baumhauer. Zeitschr. f. Kryst., 1899, *32*, 164.
Pyramids: $\epsilon(\bar{4}47)$, $e(\bar{3}31)$, $v(551)$, $h(556)$, $\tau(10.10.3)$, $(29.29.30)$, $\theta(\bar{1}\bar{1}.11.3)$, $o(111)$, $\eta(\dot{2}3.23.6)$.

NARSARSUKITE.

Tetragonal.

G. Flink. Medd. om Grönland, 1898, *14*, 234.
Base: $c(001)$.
Prisms: $a(100)$, $m(110)$, $n(210)$.
Pyramid: $p(111)$.

NATROCHALCITE.

Monoclinic.

C. Palache and C. H. Warren. Am. Jour. Sci., 1908, *26*, 345.
Pinacoids: $c(001)$, $b(010)$.
Prism: $m(110)$.
Pyramids: $p(111)$, $v(112)$, $u(221)$, $w(331)$, $q(\bar{1}11)$, $x(221)$.

NATROLITE.

F. Zambonini. Zeitschr. f. Kryst., 1901, *34*, 549.
Pyramids: $g(334)$, $k(335)$, $j(112)$, $p(361)$, $\epsilon(836)$, $\xi(16.16.17)$.

NEPTUNITE.

Monoclinic.

G. Flink. Geol. Fören. Förh., 1893, *15*, 196, and Zeitschr. f. Kryst., 1894, *23*, 346.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prism: $m(110)$.
Domes: $e(201)$, $d(\bar{3}01)$.

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Pyramids: $s(111)$, $v(221)$, $o(111)$, $u(\bar{5}12)$.

G. Flink. Medd. om Grönland, 1898, *14*, 232.
Dome: $f(\bar{1}01)$.
Pyramids: $r(\bar{2}21)$, $i(\bar{1}12)$, $x(311)$.

Wallenstrom. Geol. Fören. Förh., 1905, *27*, 149.
Pyramids: $p(\bar{3}11)$, $q(\bar{7}12)$.

W. E. Ford. Am. Jour. Sci., 1909, *27*, 235.
Pyramid: $g(\bar{2}11)$.

NEPHELITE.

E. Kaiser. Zeitschr. f. Kryst., 1899, *31*, 28.
Prism: (5160).

NICCOLITE.

A. Sachs. Ber. Akad. Berlin, 1902, 856.
Prism: $g(11\bar{2}0)$.
Pyramids: $\pi(10\bar{1}5)$, $z(17.0.\bar{1}7.10)$, $q(12.0.12.5)$, $v(4041)$, $w(30\bar{3}5)$, $s(90\bar{9}5)$, $u(3031)$, $\gamma(22.0.22.5)$, $o(5053)$, $\alpha(23.0.\bar{2}3.10)$, $t(16.0.\bar{1}6.15)$.

OCTAHEDRITE.

F. Millosevich. Rend. Acc. Linc., 1905, *14*, 92.
Second Order Pyramids: $R(3.0.10)$, $E(203)$.
First Order Pyramid: $H(332)$.
Ditetragonal Pyramids: $T(11.2.12)$, $L(25.11.5)$.

C. Hintze. Handbuch der Mineralogie, 1906, *1*, 1563.
Second Order Pyramid: $G(104)$.

R. W. Harre. Zeitschr. f. Kryst., 1906, *42*, 280.
First Order Pyramids: (5.5.43), (5.5.29).
Ditetragonal Pyramids: (45.36.50), (63.3.14), (180.3.20).

R. W. Solly. Min. Mag., 1904, *14*, 16.
First Order Pyramid: (553).
Ditetragonal Pyramids: (35.3.5), (40.3.5), (24.14.7)?.

W. Prinz. Bull. Acad. Belg., 1907, 706.
First Order Pyramid: (449).

G. Cesàro. Bull. Acad. Belg., 1907, 336.
First Order Pyramid: $M(338)$.

K. Busz. Zeitschr. f. Kryst., 1892, *20*, 529.
Ditetragonal Pyramid: $\sigma(11.3.44)$.

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Olivenite.

O. C. Farrington and E. W. Tillotson. Field Col. Mus. Geol., Ser. 3, 1908, 152.

Domes: $s(034)$, $d(025)$.

Orpiment.

(Shown to be monoclinic by Stevanović.)

S. Stevanović. Zeitschr. f. Kryst., 1904, *39*, 14.

Pinacoids: $a(100)$, $b(010)$.

Prisms: $s(320)$, $m(110)$, $u(120)$, $g(130)$, $h(160)$.

Domes: $o(101)$, (103).

Pyramids: $v(\bar{1}21)$, $v(343)$, $i(243)$, $q(\bar{4}49)$, $\chi(\bar{4}23)$, $x(323)$, $\beta(\bar{2}32)$, $f(\bar{2}52)$, $y(\bar{5}85)$, $t(21.20.\bar{3})$, $Q(\bar{2}36)$, $z(\bar{5}23)$, (143), $k(\bar{1}23)$.

O. C. Farrington and E. W. Tillotson. Field Col. Mus. Geol., Ser. 3, 1908, 154.

Domes: $d(\bar{1}03)$, $l(023)$.

Pyramid: $n(\bar{1}33)$.

Orthoclase.

A. Cathrein. Min. Mitth., 1888, *10*, 524.

Prism: $\zeta(210)$.

Domes: $\omega(017)$, $\chi(39.0.38)$.

Bartalini. Acc. Sc. Med. Nat. Ferrara, 1901.

Prisms: (310), (740), $(7.20.0)$.

Domes: $(\bar{1}3.0.12)$, $(\bar{1}6.0.13)$, $(\bar{5}06)$, $(\bar{1}4.0.15)$, $(\bar{1}2.0.1)$.

Pyramids: $(\bar{7}37)$, $(11.1.\bar{1}0)$, $(\bar{5}.5.\bar{1}1)$, $(\bar{1}4.\bar{1}4.15)$, $(\bar{9}71)$.

Paralaurionite.

Monoclinic.

G. F. Herbert Smith. Min. Mag., 1899, *12*, 108.

Pinacoids: $a(100)$, $c(001)$.

Prism: $m(110)$.

Domes: $d(101)$, $h(20\bar{1})$, $k(40\bar{1})$, $l(60\bar{1})$.

Pyramid: $p(111)$.

A. Arzruni. Zeitschr. f. Kryst., 1899, *31*, 229. (Rafaelite.*)

Domes: $(\bar{1}03)$, (101), $(20\bar{1})$, (010).

Pyramid: $(\bar{4}32)$.

* $(\bar{1}32)$ of rafaelite = (111) of paralaurionite.

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PARATACAMITE.

Hexagonal, rhombohedral.

G. F. Herbert Smith. Min. Mag., 1906, *14*, 120.

Base: $c(0001)$.

Prism: $a(11\bar{2}0)$.

Rhombohedrons: $\omega(20\bar{2}5)$, $v(7.0.\bar{7}.13)$, $w(4047)$, $r(10\bar{1}1)$, $e(01\bar{1}2)$, $f(02\bar{2}1)$.

Scalenohedron: $l(24\bar{6}1)$.

PARISITE.

G. Flink. Medd. om Grönland, 1899.

Prism: $m(10\bar{1}0)$.

Rhombohedrons: $r(2023)$, $s(4043)$.

PEARCEITE.

Monoclinic.

S. L. Penfield. Am. Jour. Sci., 1896, *2*, 17.

Pinacoids: $a(100)$, $b(010)$, $c(001)$.

Prisms: $l(310)$, $m(110)$, $h(130)$.

Domes: $k(021)$, $d(102)$, $n(101)$, $t(201)$, $e(401)$, $f(601)$, $\Delta(\bar{2}03)$, $n^{\circ}(\bar{1}01)$, $t^{\circ}(201)$, $e^{\circ}(401)$, $f^{\circ}(\bar{6}01)$.

Pyramids: $o(114)$, $r(112)$, $p(111)$, $v(332)$, $s(221)$, $u(331)$, $o^{\circ}(\bar{1}14)$, $q^{\circ}(\bar{1}13)$, $r^{\circ}(\bar{1}12)$, $p^{\circ}(111)$, $v^{\circ}(332)$, $s^{\circ}(221)$, $u^{\circ}(\bar{3}31)$, $x(311)$, $y(313)$, $z(3.1.12)$.

PECTOLITE.

A. J. Moses. Am. Jour. Sci., 1901, *12*, 99.

Prism: $h(540)$.

Domes: $y(1.0.25)$, $x(102)$.

PENFIELDITE.

Hexagonal.

S. L. Penfield. Am. Jour. Sci., 1894, *48*, 114.

Base: $c(0001)$.

Prism: $m(1010)$.

Pyramid: $p(1122)$.

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Phenacite.

S. J. Spencer. Min. Mag., 1906, *14*, 178.
Scalenohedron: $y_1(1565)$.

Piedmontite.

F. Zambonini. Zeïtschr. f. Kryst., 1902, *37*, 15.
Domes: $m_x(102)$, $(\bar{1}07)$, $l(\bar{2}01)$.

Pisanite.

W. T. Schaller. Bull. Univ. Cal. (Geol.), 1903, *3*, 199.
Pinacoid: $a(100)$.
Prisms: $h(210)$, $f(320)$, $l(120)$.
Dome: $v(101)$.
Pyramids: $r(111)$, $E(335)$, $D(221)$, $\sigma(121)$.

Pirssonite.

Orthorhombic.

J. H. Pratt. Am. Jour. Sci., 1896, *2*, 126.
Pinacoid: $b(010)$.
Prism: $m(110)$.
Pyramids: $p(111)$, $p_1(111)$, $e(131)$, $x(311)$.

Plagionite.

L. J. Spencer. Min. Mag., 1897, *11*, 192.
Pinacoid: $b(010)$.
Prism: $n(110)$.
Domes: $f(\bar{1}01)$, $g(201)$.
Pyramids: $l(223)$, $h(223)$, $t(445)$, $k(111)$.

Podolite.

Hexagonal.

W. Tschirwinsky. Centralbl. f. Min., 1907, 279.
Base: $c(0001)$.
Prism: $m(10\bar{1}0)$.

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POLYBASITE.

(Shown to be monoclinic by Penfield.)

S. L. Penfield. Am. Jour. Sci., 1896, *2*, 17.
Pinacoid: $c(001)$.
Prisms: $l(310)$, $m(110)$.
Domes: $n(101)$, $\Delta(\bar{2}03)$, $n^{\circ}(\bar{1}01)$, $\pi(\bar{4}03)$, $t^{\circ}(\bar{2}01)$.
Pyramids: $o(114)$, $r(112)$, $p(111)$, $s(221)$, $u(331)$?, $o^{\circ}(\bar{1}14)$, $r^{\circ}(\bar{1}12)$, $p^{\circ}(\bar{1}11)$.

PREHNITE.

Goodchild. Trans. Geol. Soc. Glascow, 1903, *12*, Suppl. 1.
Prism: $g(210)$.
Domes: $x(104)$, $e(012)$.

E. Billows. Rev. d. min. e crist., 1901, *27*.
Domes: (701), (501).

O. B. Böggild. Min. Grönland, 1905, 291.
Prism: (410).
Dome: (301).

PROLECTITE.

Monoclinic.

H. Sjögren. Bull. Geol. Inst. Univ. Upsala, 1894.
Pinacoids: $A(001)$, $C(010)$.
Prism: $\frac{1}{2}o(110)$.
Domes: $+\frac{1}{3}e(103)$, $-\frac{4}{9}e(\bar{4}09)$, $+\frac{5}{3}e(503)$, $i(012)$, $2i(014)$.
Pyramids: $-m(111)$, $-\frac{2}{3}n(\bar{2}23)$, $-\frac{2}{7}n(227)$, $-r(121)$, $+r(121)$, $+\frac{3}{2}r(322)$, $+\frac{3}{7}r(337)$, $-\frac{2}{9}r(229)$.

PYRARGYRITE AND PROUSTITE.

H. A. Miers. Min. Mag., 1888, *7*, 37.
Scalenohedron: $\rho(27\bar{9}7)$.

K. Busz. Zeitschr. f. Kryst., 1892, *20*, 529.
Second Order Pyramid: $p''(11\bar{2}6)$.

Lamplough. Min. Mag., 1903, *13*, 294.
Rhombohedrons: (10.0.10.1), (13.0.$\bar{1}3$.1), (0661).

PYRITE.

G. Boeris. Rev. d. min. ital., 1901, *26*, 36.
Pyritohedron: (510).

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Trapezohedrons: (855), (10.7.7).
Trisoctahedrons: (553), (554), (775).

A. Franzenau. Math. u. nat. Ber. Ungarm, 1898, *15*.
Pyritohedrons: (11.5.0), (13.8.0), (13.9.0), (11.7.0), (10.7.0), (15.11.0), (950), (970).
Diploid: (16.9.1).

K. Zimányi. Zeitschr. f. Kryst., 1899, *32*, 243.
Diploid: z(541).

K. Zimányi. Zeitschr. f. Kryst., 1904, *39*, 125 and 140.
Pyritohedrons: I(21.1.0), $\mathfrak{p}$(17.1.0), U(15.1.0), H(14.1.0), G(12.1.0), c_a(10.1.0), B(810), J(11.2.0), C(16.3.0), A(11.3.0), χ(850), Λ(11.10.0).
Diploids: $\mathfrak{a}$(11.9.7), $\mathfrak{e}$(14.11.8), $\mathfrak{b}$(852), $\mathfrak{g}$(951), $\mathfrak{h}$(13.7.1), $\mathfrak{r}$(25.15.6), $\mathfrak{w}$(7.11.22).

K. Zimányi. Földt. Közl., 1905, *35*, 544.
Trapezohedron: ζ(711).

B. Mauritz. Zeitschr. f. Kryst., 1904, *39*, 357.
Pyritohedrons: (830), (12.5.0), (17.14.0).
Trapezohedron: (15.14.14).
Diploids: (821), (631), (18.9.2), (15.11.7), (11.8.5), (10.5.2), (45.36.20), (24.15.10), (11.7.5).

B. Mauritz. Földt. Közl., 1904, *34*.
Diploids: $\mathfrak{A}$(18.10.5), $\mathfrak{D}$(654), $\mathfrak{B}_1$(456).

K. Busz. Neues Jahrb. Min., 1901, *2*.
Trapezohedron: (11.3.3).

C. Travis. Proc. Am. Phil. Soc., 1906, *45*, 131.
Trapezohedron: (744).
Trisoctahedrons: (11.11.4), (552), (773), (774), (443), (665).
Diploids: (753), (952), (14.9.3), (876), (12.11.10).

V. Goldschmidt and H. Philipp. Zeitschr. f. Kryst., 1902, *36*, 386.
Diploids: x(213), O(325), Γ(215).

V. Rosický. Abh. d. böhm. Akad., 1903.
Diploid: C(964).

J. Samojloff. Mater. z. geol. Russl., 1906, *23*.
Diploid: (10.9.3).

E. H. Kraus and I. D. Scott. Zeitschr. f. Kryst., 1907, *44*, 144.
Pyritohedrons: e_b(16.5.0), e_c(13.10.0).

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PYROAURITE.

G. Flink. Bull. Geol. Inst. Univ. Upsala, 1900, *5*, 81.
Base: $c(0001)$.
Rhombohedrons: $g(40\bar{4}1)$, $r(10\bar{1}1)$, $f(02\bar{2}1)$.

PYROMORPHITE.

A. Lacroix. Bull. soc. fr. min., 1894, *17*, 120.
Pyramids: $(15.0.\bar{1}\bar{5}.4)$, $(90\bar{9}1)$.

PYROXENE.

V. v. Zepharovich. "Lotos," 1889.
Prism: (350).
A. Cathrein. Ann. d. K. K. nat. Hoffmus., 1889, *4*, 181.
Dome: (302).
Pyramids: $(\bar{4}41)$, $(\bar{1}.2.10)$.
K. Busz. Zeitschr. f. Kryst., 1892, *20*, 529.
Pyramid: $h'(551)$.
A. Schmidt. Zeitschr. f. Kryst., 1892, *21*, 1.
Prisms: $\mathfrak{F}(10.1.0)$, $\mathfrak{G}(710)$, $\mathfrak{H}(750)$, $\mathfrak{K}(140)$, $\mathfrak{M}(160)$.
Dome: $\mathfrak{N}(\dot{0}.11.5)$.
Pyramids: $\mathfrak{S}(\bar{4}14)$, $\mathfrak{V}(421)$, $\mathfrak{W}(\bar{5}31)$.
F. Zambonini. Zeitschr. f. Kryst., 1900, *33*, 39, and 1901, *34*, 225.
Pyramids: $\mathfrak{v}(\bar{3}43)$, $\mathfrak{V}(\bar{1}61)$, $\mathfrak{w}(14.1.1)$, $F(\bar{3}61)$, $G(\bar{3}.12.1)$.
F. Zambonini. Zeitschr. f. Kryst., 1904, *40*, 52.
Dome: $\mathfrak{V}(605)$.
Iwasak. Zeitschr. f. Kryst., 1900, *32*, 302.
Dome: (203).

QUARTZ.

O. Luedecke. Min. d. Harzes, 1896, 196.
Rhombohedrons: $(0.13.\bar{1}\bar{3}.7)$, $(0.13.\bar{1}\bar{3}.6)$, $(16.0.\bar{1}\bar{6}.3)$, $(13.0.\bar{1}\bar{3}.10)$, $(\bar{1}\bar{6}.0.\bar{1}\bar{6}.13)$.
A. Cathrein. Zeitschr. f. Kryst., 1889, *17*, 19.
Rhombohedrons: $(70\bar{7}5)$.
Trigonal Trapezohedrons: $(11.1.\bar{1}\bar{2}.12)$, $(1.11.\bar{1}\bar{2}.12)$, $(8.1.\bar{9}.10)$, $(92\bar{7}8)$.
D. Rossignoli. Riv. d. min. e. crist. ital., 1891, *10*, 3.
Rhombohedrons: $(70\bar{7}6)$, $(14.0.\bar{1}\bar{4}.3)$, $(0.23.\bar{2}\bar{3}.16)$, (0553).

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Trigonal Trapezohedrons: (2752), (2572), (5.12.7.5), (5.7.12.5), (2.14.12.11), (1.13.12.10), (1.12.13.10), (11.1.12.11).

F. Gonnard. Bull. Soc. fr. Min., 1899, *22*, 94.

Rhombohedrons: (41.0.41.11), (8085), (0.11.11.10), (0.11.11.7), (0.19.19.11).

Trigonal Trapezohedrons: (13.3.16.16), (32.15.47.47), (23.15.38.38), (4377), (13.12.25.25), (5388), (8.15.23.23), (2577), (1677), (2.15.17.17), (1.15.16.16), (1.30.29.29), (22.3.25.22), (11.6.17.11), (26.15.41.26), (8.5.13.8), (41.30.71.41), (19.15.34.19).

J. Samojloff. Mater z. geol. Russl., 1906, *23*.

Rhombohedron: (20.0.20.1).

Trigonal Trapezohedron: (5.6.11.5).

E. Artini. Rend. Accad. d. Linc., 1888, (4), *5*.

Trigonal Trapezohedrons: (7.15.22.7), (7.22.15.7), (3473), (3.743), (4594), (4954), (8.9.17.8), (8.17.9.8), (7.6.13.7), (13.6.7.7), (8.5.13.8), (13.5.8.8), (7187), (8177), (1.13.12.10), (1.12.13.10), (2.14.12.11), (2.12.14.11), (4.20.24.21).

G. Lincio. Neues Jahrb. f. Min., 1903, Beil.-B., *18*, 155.

Rhombohedrons: $V(10\bar{1}5)$, y:(3035)?, $w(03\bar{3}4)$, $x(40\bar{4}5)$, x:(04$\bar{4}$5), $\omega(90\bar{9}7)$.

Trigonal Trapezohedrons: W.(81$\bar{9}$8), V:(15.8.23.15), U:(9.$\bar{5}$.14.9), T:(12.7.19.12), L:(3$\bar{2}$53), K:(4374), α:(9.1.10.11), Q:(61$\bar{7}$9), ζ:(7.2.9.16), c:(9.3.12.7)?, b:(6.3.9.13), β:(4.3.7.11), T:(11.8.19.12).

RASPITE.

Monoclinic.

C. Hlawatsch. Zeitschr. f. Kryst., 1897, *29*, 137, and 1906, *42*, 587.

Pinacoids: $a(100)$, $b(010)$, $c(001)$.

Prism: $m(110)$.

Domes: $e(101)$, $f(\bar{1}02)$, $d(011)$.

Pyramids: $p(122)$, $\delta(1.12.12)$.

RATHITE.

Orthorhombic (Solly's orientation).

H. Baumhauer. Zeitschr. f. Kryst., 1896, *26*, 593.

Pinacoid: $b(010)$.

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Prisms: $9r(190)$, $8r(180)$, $2\frac{2}{3}r(3.22.0)$, $2\frac{0}{3}r(3.20.0)$, $6r(160)$, $1\frac{7}{3}r(3.17.0)$, $1\frac{4}{3}r(3.14.0)$, $4r(140)$, $1\frac{1}{3}r(3.11.0)$, $1\frac{0}{3}r(3.10.0)$, $\frac{8}{3}r(380)$, $\frac{9}{4}r(490)$, $2r(120)$, $\frac{5}{3}r(350)$, $\frac{4}{3}r(340)$, $r(110)$, $\frac{8}{7}s(870)$, $\frac{3}{2}s(320)$, $3s(310)$, $\frac{9}{2}s(920)$.

Domes: $\frac{7}{3}k(073)$, $\frac{5}{3}k(053)$, $\frac{10}{11}k(0.10.11)$, $\frac{1}{3}k(013)$.

R. H. Solly. Min. Mag., 1901, *13*, 77.

Pinacoids: $a(100)$, $c(001)$.

Prisms: $\frac{80}{3}r(3.80.0)$, $18r(1.18.0)$, $11r(1.11.0)$, $10r(1.10.0)$, $7r(170)$, $\frac{16}{3}r(3.16.0)$, $3r(130)$, $\frac{5}{2}r(250)$, $\frac{7}{3}r(370)$, $\frac{7}{4}r(470)$, $2s(210)$, $4s(410)$, $7s(710)$, $8s(810)$.

Domes: $h(101)$, $15k(0.15.1)$, $7k(071)$, $5k(051)$, $\frac{7}{2}k(072)$, $3k(031)$, $\frac{8}{3}k(083)$, $\frac{9}{5}k(095)$, $\frac{7}{4}k(074)$, $\frac{4}{3}k(043)$, $k(011)$, $\frac{5}{6}k(056)$.

Pyramids: $\frac{11}{3}q(3.11.3)$, $3q(131)$, $\frac{5}{2}q(252)$, $\frac{7}{3}q(373)$, $\frac{5}{3}q(353)$, $p(111)$, $3t(313)$, $5t(515)$, $2p(211)$, $2m(122)$, $\frac{3}{2}w(132)$.

REALGAR.

(New orientation.)

C. Vrba. Zeitschr. f. Kryst., 1889, *15*, 449.

Prism: $\zeta(250)$.

G. Grattarola. Gior. d. Min., 1890, *1*, 278.

Pyramid: $(\bar{3}13)$?.

V. Hackmann. Zeitschr. f. Kryst., 1896, *27*, 608.

Prism: $\zeta(450)$.

RETZIAN.

Orthorhombic.

H. Sjögren. Bull. Geol. Inst. Upsala, 1894, *2*, 54.

Pinacoid: $b(010)$.

Prisms: $m(110)$, $n(130)$.

Domes: $d(101)$, $k(071)$.

RHODONITE.

A. Hamberg. Geol. Fören. Förh., 1891, *13*, 539.

Dome: $\theta(0\bar{1}3)$.

Pyramid: (423).

L. V. Pirsson. Am. Jour. Sci., 1890, *40*, 484.

Dome: $\epsilon(201)$.

C. Anderson. Rec. Austr. Mus., 1908, *7*, 129.

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Domes: $B(0\bar{1}5)$, $C(0\bar{1}4)$, $A(013)$, $E(037)$, $D(013)$, $F(103)$, $G(207)$, $H(201)$.
Pyramids: $K(\bar{1}\bar{1}3)$, $L(\bar{1}\bar{1}2)$.

RHÖNITE.

Triclinic.

J. Soellner. Neues Jahrb. f. Min., 1907, Beil.-B., *24*, 475.
Pinacoids: $b(010)$, $a(100)$, $c(001)$.
Prisms: $m(110)$, $\mu(1\bar{1}0)$.
Pyramids: $r(\bar{1}\bar{1}1)$, $k(111)$, $v(131)$, $i(\bar{1}31)$.

RIEBECKITE.

Monoclinic.

W. J. Sollas. Proc. R. Irish Acad., 1895, *3*, 516.
Pinacoid: $b(010)$.
Prisms: $m(110)$, $x(150)$.
Domes: $t(101)$, $p(\bar{1}01)$, $r(011)$.
Pyramid: $z(\bar{1}21)$.

SAFFLORITE.

Orthorhombic.

H. Sjögren. Bull. Geol. Inst. Upsala, 1894, *2*, 68.
Pinacoids: $a(100)$, $c(001)$.
Prism: $m(110)$.
Dome: $d(011)$.
Pyramid: $o(111)$.

SARTORITE.

Monoclinic.

C. O. Trechmann. Min. Mag., 1907, *14*, 212.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $\frac{9}{2}f(920)$, $4f(410)$, $\frac{5}{2}f(520)$, $2f(210)$, $\frac{11}{4}f(11.7.0)$, $\frac{3}{2}f(320)$, $f(110)$.
Domes: $\alpha(023)$, $\kappa(045)$, $i(011)$, $w(041)$, $\frac{9}{2}d(\bar{9}02)$, $4d(401)$, $2d(\bar{2}01)$, $d(101)$, $-\frac{4}{3}d(403)$, $-d(\bar{1}01)$.
Pyramids: $e(111)$, $\gamma(441)$, $\rho(113)$, $\Delta(445)$, $r(\bar{1}11)$, $m(443)$, $z(221)$, $o(\bar{4}41)$, $\epsilon(\bar{2}45)$, $h(122)$, $g(243)$, $\beta(\bar{1}21)$, $v(241)$, $p(\bar{1}22)$, $\mu(243)$, $q(241)$, $x(211)$, $\nu(\bar{2}11)$, $D(641)$, $s(643)$, $t(321)$, $\tau(322)$, $n(\bar{3}22)$.

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SCHEELITE.

F. Zambonini. Rend. Accad. Linc., 1906, (5), *15*, 558.
Second Order Pyramids: $\lambda(507)$, $\nu(407)$?.
First Order Pyramids: $\mu(338)$, $\zeta(227)$.
L. Colomba. Rend. Accad. Linc., 1906, (5), *15*, 281.
First Order Pyramid: (885).
Third Order Pyramids: (323) ?, (21.1.11) ?, (714), (735), (756), (232).

SCHIZOLITE.

Triclinic.

O. B. Böggild. Medd. om Grönl., 1903, *26*, 93.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prisms: $o(530)$, $m(110)$, $P(230)$, $M(1\bar{1}0)$, $l(120)$.
Domes: $r(\bar{1}02)$, $n(\bar{1}01)$, $s(\bar{2}01)$.
Pyramids: $e(111)$, $g(\bar{1}11)$, $f(\bar{1}\bar{4}1)$.

SCOLECITE.

G. Flink. Bihang. K. Ver. Akad. Handl., *13*, (2), 8.
Prisms: (510), (470).
Pyramids: (441), (331), (12.12.5), (474), (551).

SCORODITE.

K. Busz. Zeitschr. f. Kryst., 1892, *20*, 555.
Dome: $f(011)$.

SELIGMANNITE.

Orthorhombic.

H. Baumhauer. Ber. Ak. Berlin, 1901, 110.
Pinacoids: (100), (010), (001).
Prisms: $A(410)$, (110), (120).
Domes: $x(013)$, $z(021)$, $\Sigma(031)$, $B(071)$, (011), (101).
Pyramids: $C(311)$, $O(1.10.2)$.
R. H. Solly. Min. Mag., 1903, *13*, 336, and 1906, *14*, 184.
Prisms: $E(610)$, $\eta(310)$, $l(320)$.
Domes: $\frac{2}{5}x(025)$, $F(061)$, $\Delta(105)$, $t(104)$, $\epsilon(103)$, $x(102)$, $h(203)$, $I(201)$, $H(703)$, $G(601)$.
Pyramids: $\phi(113)$, $\frac{2}{9}p(229)$, $3p(331)$, $4p(441)$, $O(313)$,

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$S(212)$, $L(131)$, $K(161)$, $N(323)$, $M(233)$, $D(322)$, $R(533)$, $Q(733)$, $P(611)$, $Z(261)$, $Y(312)$, $U(413)$, $T(613)$, $S(713)$, $V(12.1.2)$, $X(14.3.6)$.

Senaite.

Hexagonal-trirhombohedral.

E. Hussak and G. T. Prior. Min. Mag., 1898, *12*, 30.
Base: $c(0001)$.
Rhombohedrons: $r(10\bar{1}1)$, $S(20\bar{2}1)$, $s(40\bar{4}1)$.

Siderite.

F. Gonnard. Bull. soc. fr. Min., 1895, *18*, 382.
Rhombohedrons: $(03\bar{3}2)$, $(10\bar{1}2)$, $(30\bar{3}4)$.
G. Cesàro. Ann. Soc. géol. Belg., 1891, *18*.
Scalenohedron: $(24\bar{6}1)$.
W. T. Schaller. Am. Jour. Sci., 1906, *21*, 364.
Rhombohedrons: $l(70\bar{7}5)$, $k(50\bar{5}2)$.
Scalenohedron: $y(32\bar{5}1)$.
M. Henglein. Zeitschr. f. Kryst., 1907, *43*, 575.
Rhombohedrons: $l(3031)$, $\Delta(70\bar{7}2)$.

Sillimanite.*

E. Taubert. Centralbl. f. Min., 1906, 372.
Dome: (052).

Skutterudite.

L. Staudenmair. Zeitschr. f. Kryst., 1892, *20*, 468.
Pyritohedron: $e(210)$.

Smithite.

Monoclinic.

R. H. Solly. Min. Mag., 1905, *14*, 72.
Pinacoids: $a(100)$, $c(001)$.
Domes: $e(101)$, $d(10\bar{1})$, (011).
Pyramids: (411), $r(311)$, $q(211)$, (322), $p(111)$, (355), (511), $(\bar{4}11)$, $Q(\bar{2}11)$, $P(111)$, $(\bar{2}12)$.
G. F. Herbert Smith and G. T. Prior. Min. Mag., 1907, *14*, 293.

* New orientation; $a:b:c$ (Taubert) $= c:b:a$ (Goldschmidt).

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Prism: $l(320)$.
Dome: $h(102)$.
Pyramid: $R(\bar{3}11)$.

SMITHSONITE.

H. Buttgenbach. Bull. soc. fr. Min., 1906, *29*, 190.
Rhombohedron: $(01\bar{1}5)$.

SODALITE.

F. Zambonini. Rend. Acc. Linc., 1906, *15*, 235.
Hexoctahedron: (321).

SPERRYLITE.

Isometric-pyritohedral.

S. L. Penfield. Am. Jour. Sci., 1889, *37*, 71.
Cube: $a(100)$.
Dodecahedron: $d(110)$.
Octahedron: $o(111)$.
Pyritohedron: $e(210)$.
T. L. Walker. Am. Jour. Sci., 1896, *1*, 110.
Diploid: $A(5.2.10)$?.
V. Goldschmidt and W. Nicol. Am. Jour. Sci., 1903, *15*, 450.
Pyritohedrons: $a(310)$, $g(520)$, $h(530)$, $b(320)$.
Trapezohedrons: $k(411)$, $m(311)$, $q(211)$, $B(533)$.
Trisoctahedron: $u(221)$.
Diploids: $\psi(421)$, $x(321)$, $z(531)$?, $D(632)$?.

SPHALERITE.

G. Cesàro. Bull. Acad. Belg., 1893, *25*, 88, and Ann. Soc. Géol. Belg., 1890, *17*, 237.
Hexakistetrahedrons: (316), $(6\bar{1}8)$.
A. Franzenau. Zeitschr. f. Kryst., 1896, *27*, 95.
Tetragonal Tristetrahedron: (10.10.1).
W. H. Hobbs. Bull. Univ. Wis., 1895, *1*, 134.
Tetragonal Tristetrahedron: (775).
O. Luedecke. Min. d. Harzes, 1896, 53.
Tetragonal Tristetrahedron: $e^2(55\bar{3})$.
Trigonal Tristetrahedron: $t^2(7\bar{3}3)$.

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A. F. Rogers. Am. Jour. Sci., 1900, *9*, 134.
Trigonal Tristetrahedron: $\sigma(833)$?.
A. Mühlhauser. Min. petr. Mitth., 1901, *20*, 83.
Tetragonal Tristetrahedron: $y(441)$.
L. Colomba. Rend. Acad. Linc., 1906, *15*, 640.
Hexakistetrahedron: (632).

Spodiosite.

Orthorhombic.

G. Nordenskiöld. Geol. Fören. Förh., 1893, *15*, 460.
Pinacoids: $a(100)$, $b(010)$, $c(001)$.
Prism: $m(110)$.
Domes: $d(102)$, $e(021)$.
Pyramids: $p(111)$, $q(254)$?, $r(854)$?.

Spodumene.

W. T. Schaller. Univ. Calif. Bull. (Geol.), 1903, *3*, 265.
Prism: $A(350)$.

Stannite.

Tetragonal-sphenoidal.

L. J. Spencer. Min. Mag., 1901, *13*, 54.
Base: $c(001)$.
Prisms: $a(100)$, $m(110)$.
Second Order Pyramids: $e(101)$, $z(201)$.
Sphenoids: $d(114)$, $n(112)$, $p(111)$, $t(221)$, $-n(112)$, $-p(1\bar{1}1)$.
Scalenohedron: $u(423)$.

Steenstrupine.

Hexagonal-rhombohedral.

J. C. Moberg. Zeitschr. f. Kryst., 1898, *29*, 386.
Base: $c(0001)$.
Prism: $a(1120)$.
Rhombohedrons: $\rho(5059)$, $r(10\bar{1}1)$, $z(40\bar{4}1)$, $\epsilon(01\bar{1}3)$, $e(01\bar{1}2)$, $f(04\bar{4}5)$, $d(02\bar{2}1)$, $\xi(08\bar{8}1)$.

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STELZNERITE.

Orthorhombic.

A. Arzuni and K. Thaddéeff. Zeitschr. f. Kryst., 1899, *31*, 229.
Pinacoids: $b(010)$, $c(001)$.
Prism: $m(110)$.
Dome: $o(011)$.
Pyramid: $p(111)$.

STEPHANITE. (Dana's orientation.)

E. Artini. Giorn. d. Min., 1891, *2*, 241.
Prisms: (510)?, (230).
Dome: (0.5.11).
Pyramids: (818), (18.5.5), (13.4.4), (372), (7.11.9), (141), (2.10.1), (161).

Nejdl. Ber. Böhm. Ges., 1895.
Pyramids: $N(522)$, $r(44\bar{1})$, $h_2(9.13.18)$?, $t_3(8.33.16)$?.

L. J. Spencer. Min. Mag., 1897, *11*, 192.
Pyramids: (551), (10.10.13).

F. Slavic. Abh. böhm. Akad. Prag., 1901, *16*.
Prisms: (340), (3.10.0), (140), (160).
Domes: (087), (038).
Pyramids: (883), (553), (11.11.5), (17.17.3), (5.17.9).

G. d'Achiardi. Att. Soc. Tosc., 1901, Mem., *18*.
Domes: (061), (041), (027), (038), (035), (056).
Pyramids: (776), (885), (772), (551), (485), (1.3.19), (1.3.17), (1.4.23), (1.4.20), (1.5.30), (3.27.7).

STIBIOTANTALITE.

Orthorhombic-hemimorphic.

S. L. Penfield and W. E. Ford. Am. Jour. Sci., 1906, *22*, 61.
Pinacoids: $a(100)$, $a'(\bar{1}00)$.
Prisms: $m(110)$ or $(\bar{1}10)$?, $q(130)$, $q'(\bar{1}30)$.
Domes: $\eta(209)$, $\eta'(\bar{2}09)$, $h(203)$, $h'(\bar{2}03)$?, $\delta(043)$.
Pyramids: $w(4.12.9)$, $w'(\bar{4}.12.9)$.

STIBNITE.

A. Schmidt. Zeitschr. f. Kryst., 1898, *29*.
Pyramids: $\mathfrak{s}(40.19.10)$, $\mathfrak{A}(10.9.15)$, $\mathfrak{r}(563)$, $\mathfrak{w}(12.19.3)$.

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H. Ungemach. Bull. soc. fr. Min., 1906, *29*, 264.
Prisms: (380), (180).

STOKESITE.

Orthorhombic.

A. Hutchinson. Min. Mag., 1900, *12*, 274.
Pinacoids: b(010), c(001).
Prism: m(110).
Pyramids: s(565), t(122), v(121).

STOLZITE.

B. K. Emerson. U. S. Geol. Surv., 1895, Bull. *126*, 163.
Prisms: (120), (130).
Third Order Pyramids: (131), (342).
C. Hlawatsch. Zeitschr. f. Kryst., 1897, *29*, 130.
Prism: a(100).
Second Order Pyramids: Ω(1.0.10)?, ω(109), τ(103), o(102), η(203), h(304), ϵ(201).
E. Artini. Rend. Ist. Lomb. Milano, 1905, *38*, 373.
First Order Pyramids: (113), (115), (117), (119).
Third Order Pyramids: (233), (344)?, (455)?, (899)?.

STRONTIANITE.

C. Vrba, Zeitschr. f. Kryst., 1889, *15*, 449.
Dome: v(031).
A. Cathrein. Zeitschr. f. Kryst., 1888, *14*, 366.
Pyramids: d(661), n(115).
J. Beykirch. Neues Jahrb. f. Min., 1901, Beil.-B., *13*, 389.
Domes: f(051), μ(071)?, ν(0.11.1).
Pyramids: r(11.11.4)?, Θ(10.10.1), π(24.24.1), Ψ(36.36.1).

STRÜVERITE.

Tetragonal.

G. T. Prior and F. Zambonini. Min. Mag., 1908, *15*, 78.
Prisms: a(100), m(110).
Pyramid: s(111).

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STYLOTYPITE.

Monoclinic.

S. Stevanović. Zeitschr. f. Kryst., 1902, *37*, 235.
Pinacoid: $a(100)$.
Prisms: $\mu(310)$, $n(210)$, $m(110)$.
Domes: $r(101)$, $t(302)$, $s(401)$, $d(032)$.
Pyramids: $x(111)$, $y(332)$, $q(313)$, $o(311)$.

SULPHOBORITE.

Orthorhombic.

H. Bücking. Ber. Ak. Berlin, 1893, 967.
Pinacoids: $b(010)$, $c(001)$.
Prism: $m(110)$.
Dome: $r(101)$.
Pyramid: $o(111)$.

SULPHUR.

F. Millosevich. Riv. Min. Ital., 1898, *21*, 43.
Dome: $\phi(305)$.
Pyramids: $\lambda(155)$, $\mu(319)$.
K. Busz. Zeitschr. f. Kryst., 1892, *20*, 529.
Pyramids: $f(151)$, $\eta(553)$.
A. Pelikan. Min. petr. Mitth., 1892, *12*, 344.
Pyramid: $k_0(122)$.
H. Buttgenbach. Ann. Soc. géol. Belg., 1898, *25*, 73.
Domes: (201), (102), (043).
Pyramids: (227), (229) ?, (115), (3.3.16), (116).

SYLVANITE.

W. H. Hobbs. Am. Jour. Sci., 1899, *7*, 357. (Goldschmidtite.*)
Domes: $X(403)$, $q(203)$.
Pyramids: $t(292)$, $l(161)$.
C. Palache. Am. Jour. Sci., 1900, *10*, 419.
Pyramids: $v(525)$, $w(343)$, $j(521)$, $u(723)$.
A. J. Moses. Am. Jour. Sci., 1905, *20*, 282.
Domes: $H_1(102)$, $T(103)$, $L(203)$.

* Determined by Palache to be Sylvanite. 100 (Goldschmidtite) = 101 (Sylvanite).

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C. Vrba. Zeitschr. f. Kryst., 1908, *44*, 69.
Prisms: $L(320)$, $U(610)$.
Dome: $G(\bar{3}02)$.
Pyramids: $T(\bar{3}12)$, $\eta(523)$, $\epsilon(\bar{5}43)$.

SYNCHISITE.

Hexagonal-rhombohedral-hemimorphic.

O. B. Böggild. Medd. om Grönland, 1906, *33*, 99.
Base: $c(0001)$.
Prisms: $a(11\bar{2}0)$, $m(10\bar{1}0)$.
Second Order Pyramids: $z(11\bar{2}1)$, $o(11\bar{2}3)$.
Rhombohedrons: $i(01\bar{1}5)$, $u(02\bar{2}9)$, $q(01\bar{1}2)$, $\beta(0332)$, $y(03\bar{3}4)$, $p(01\bar{1}1)$, $j(03\bar{3}1)$, $t(20\bar{2}9)$, $\rho(20\bar{2}3)$, $v(30\bar{3}4)$, $s(40\bar{4}3)$, $\alpha(30\bar{3}2)$.

TAINIOLITE.

Monoclinic.

G. Flink. Medd. om Grönl., 1898, *14*, 234, and 1901, *24*, 115.
Pinacoids: $b(010)$, $c(001)$.
Domes: $\theta(027)$, $e(023)$.
Pyramid: $\mu(\bar{1}11)$.

TANTALITE.

C. Vrba. Zeitschr. f. Kryst., 1889, *15*, 194.
Pyramids: $\gamma(611)$, $d(311)$, $w(344)$, $\sigma(133)$.
S. L. Penfield (see Blake). Am. Jour. Sci., 1891, *41*, 403.
Prism: $d(730)$.

TARBUTTITE.

Triclinic.

L. J. Spencer. Min. Mag., 1908, *15*, 22.
Pinacoids: $c(001)$, $b(010)$, $a(100)$.
Prism: $m(110)$.
Domes: $f(\bar{1}01)$, $s(\bar{1}02)$, $t(\bar{1}03)$, $l(021)$, $h(\bar{0}21)$, $u(0\bar{1}1)$.
Pyramids: $e(\bar{2}21)$, $g(211)$, $d(\bar{2}23)$, $k(\bar{1}11)$, $i(\bar{1}\bar{2}2)$, $r(243)$, $o(1\bar{2}\bar{1})$.

TEALLITE.

Orthorhombic?.

G. T. Prior. Min. Mag., 1904, *14*, 21.
Pinacoids: $c(001)$, $a(100)$?.

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Domes: $d(101)$?, $e(201)$?.
Pyramids: $o(111)$, $p(221)$, $t(211)$?.

TERLINGUAITE.

Monoclinic.

A. J. Moses.* Am. Jour. Sci., 1903, *16*, 225.
Pinacoids: $c(001)$, $b(010)$, $a(100)$.
Prisms: $m(110)$, $\delta(210)$.
Domes: $d(011)$, $f(031)$, $h(051)$, $t(102)$, $y(101)$, $w(301)$, $n(\bar{1}02)$, $u(\bar{1}01)$, $x(\bar{2}01)$, $z(\bar{3}01)$.
Pyramids: $s(311)$, $q(315)$, $\alpha(\bar{3}13)$, $o(\bar{3}11)$, $v(355)$, $p(111)$, $r(499)$, $i(211)$, $e(\bar{1}11)$, $l(\bar{4}33)$, $g(\bar{2}33)$, $\gamma(\bar{4}33)$, $k(334)$, $\pi(112)$, $\lambda(115)$, $\beta(\bar{1}15)$.

W. F. Hillebrand and W. T. Schaller. U. S. Geol. Surv., 1909, Bull. *405*, 83.
Prisms: $j(120)$, $\mathfrak{U}(230)$, $B(320)$, $\mathfrak{w}(520)$, $\mathfrak{b}(830)$, $\mathfrak{e}(610)$.
Domes: $D(031)$, $\mathfrak{f}(045)$, $\mathfrak{g}(035)$, $\mathfrak{h}(012)$, $\mathfrak{a}(049)$, $\mathfrak{b}(025)$, $\mathfrak{l}(017)$, $\mathfrak{n}(105)$, $\mathfrak{o}(104)$, $\mathfrak{p}(103)$, $\mathfrak{i}(308)$, $\mathfrak{r}(203)$, $\mathfrak{N}(304)$, $\mathfrak{f}(403)$, $F(503)$, $G(401)$, $\eta(901)$, $\mathfrak{k}(12.0.1)$, $M(\bar{1}07)$, $L(\bar{1}04)$, $\mathfrak{W}(\bar{1}03)$, $\mathfrak{r}(\bar{3}05)$, $N(\bar{2}03)$, $\mathfrak{y}(\bar{3}04)$, $R(\bar{4}05)$, $\Pi(\bar{6}05)$, $P(\bar{4}03)$, $Q(\bar{3}02)$, $\mathfrak{B}(\bar{5}03)$, $\mathfrak{C}(\bar{5}02)$, $S(\bar{4}01)$, $\theta(\bar{5}01)$, $\mathfrak{z}(\bar{7}01)$, $\mathfrak{F}(12\bar{0}.0.1)$.
Pyramids: $\mathfrak{c}(317)$, $U(3.1.\bar{1}1)$, $V(\bar{3}19)$, $\mathfrak{G}(1.10.\bar{1}0)$, $\tau(\bar{1}88)$, $\mathfrak{H}(\bar{1}66)$, $\mathfrak{q}(\bar{1}55)$, $\zeta(\bar{1}44)$, $\epsilon(\bar{1}33)$, $Z(\bar{1}22)$, $Y(\bar{2}33)$, $\mathfrak{K}(\bar{3}44)$, $\chi(\bar{3}22)$, $\mathfrak{M}(\bar{5}33)$, $\mathfrak{O}(\bar{4}11)$, $\sigma(\bar{1}\bar{5}5)$, $\phi(\bar{1}44)$, $X(\bar{1}33)$, $\mathfrak{L}(\bar{2}55)$, $\psi(\bar{1}22)$, $\omega(\bar{3}55)$, $\Delta(\bar{2}33)$, $\Gamma(\bar{3}44)$, $\mathfrak{Y}(\bar{6}77)$, $\mathfrak{R}(\bar{3}22)$, $\mathfrak{S}(\bar{5}22)$, $\mathfrak{T}(\bar{3}26)$, $\mathfrak{P}(\bar{3}28)$, $\mathfrak{V}(\bar{3}37)$, $O(\bar{1}33)$, $8(3.4.\bar{1}4)$, $\mathfrak{Z}(\bar{1}24)$, $\mathfrak{X}(\bar{1}26)$, $\Phi(\bar{1}28)$, $\Lambda(\bar{1}31)$, $W(\bar{6}32)$, $\Theta(\bar{2}12)$, $T(\bar{2}13)$, $\mathfrak{Y}(\bar{2}15)$, $\mathfrak{Z}(\bar{3}24)$, $\Xi(\bar{3}34)$, $\rho(\bar{1}\bar{1}3)$, $H(\bar{1}14)$, $\beta(\bar{1}15)$, $\iota(\bar{1}24)$, $I(\bar{1}31)$, $J(6.1.\bar{1}5)$, $K(\bar{6}31)$, $\mu(\bar{6}32)$, $\mathfrak{m}(\bar{6}35)$, $\Omega(\bar{2}12)$, $\xi(\bar{2}13)$, $\mathfrak{t}(\bar{2}15)$, $\mathfrak{u}(\bar{2}16)$, $A(\bar{2}17)$, $C(\bar{4}13)$, $\mathfrak{D}(\bar{4}15)$, $\mathfrak{A}(4.1.\bar{1}1)$, $\mathfrak{E}(\bar{4}35)$, $E(\bar{5}13)$, $\Upsilon(\bar{5}15)$, $\mathfrak{U}(\bar{5}19)$, $\mathfrak{Q}(\bar{5}24)$, $\mathfrak{F}(\bar{8}14)$.

TETRAHEDRITE.

C. O. Trechmann. Min. Mag., ——, *10*, 220.
Trigonal Tristetrahedrons: (755), (855), (17.10.10), (955),

* Schaller's orientation. hkl (Moses) = $3h.k.l$ (Schaller).

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(19.10.10), (21.10.10), (12.5.5), (522), (13.5.5), (27.10.10), (14.5.5), (16.5.5), (722), (19.5.5), (922), (47.10.10), (511), (11.2.2), (13.2.2), (711), (37.5.5), (811), (911), (10.1.1), (12.1.1), (16.1.1), (28.1.1), (34.1.1), (76.1.1), (28.$\bar{1}$.1).

Tetragonal Tristetrahedrons: (885), (994), (552), (12.12.1), (30.30.1), (8$\bar{8}$5), (5$\bar{5}$2), (3$\bar{3}$1), (4$\bar{4}$1), (13.$\bar{1}$3.2).

Hexakistetrahedron: (23.12.11).

H. Baumhauer. Zeitschr. f. Kryst., 1893, *21*, 202.

Trigonal Tristetrahedron: (19.6.6).

A. Cathrein. Min. petr. Mitth., 1888, *10*, 524.

Trigonal Tristetrahedron: *j'*(17.9.9).

A. Brumlechner. Zeitschr. f. Kryst., 1895, *24*, 628.

Trigonal Tristetrahedron: (21.20.20).

Tetragonal Tristetrahedrons: (7$\bar{7}$1), (11.$\bar{1}$1.2).

K. Zimányi. Zeitschr. f. Kryst., 1901, *34*, 78.

Trigonal Tristetrahedron: π(655).

H. Ungemach. Bull. Soc. fr. Min., 1906, *29*, 194.

Tetrahexahedrons: (530), (410).

Trigonal Tristetrahedrons: (988), (766), (13.10.10), (533), (744), (13.7.7), (15.8.8), (11.5.5), (944), (17.5.5), (18.5.5), (21.5.5), (22.5.5), (23.5.5), (17.3.3)?, (29.5.5), (31.5.5).

Tetragonal Tristetrahedrons: (18.18.17), (13.13.10), (443), (775), (995), (11.11.5), (773), (881)?, (991), (14.14.1)?, (35.35.1), (62.62.1)?, (26.2$\bar{6}$.25), (29.2$\bar{9}$.25), (66$\bar{5}$), (88$\bar{1}$).

Hexakistetrahedron: (532).

THALÉNITE(?).

Monoclinic.

C. Benedicks. Geol. Fören. Förh., 1898, *20*, 308.

Pinacoids: *a*(100), *b*(010), *c*(001).

Prism: *m*(110).

Dome: *f*(021).

Pyramids: *e*(111), *d*(1$\bar{1}$1), *h*(311).

THENARDITE.

A. Pelikan. Min. petr. Mitth., 1891, *12*, 476.

Prism: *u*(130).

Pyramid: *v*(113).

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THOMSONITE.

O. B. Böggild. Medd. om Grönl., 1905, *28*, 109.
Dome: (601).
G. Cesàro. Bull. Ac. Belg., 1907, 334.
Prism: (9.10.0).

THORIANITE.

Isometric.

W. R. Dunstan and G. S. Blake. Proc. Roy. Soc., 1905, A, *74*, 253.
Cube: (100).

TITANITE.

G. Flink. Geol. Fören. Förh., 1893, *15*.
Prism: (560).
Domes: ($\bar{3}02$), ($\bar{1}04$).
Pyramids: ($\bar{4}31$), ($\bar{1}39$).
P. Ternier. Bull Soc. fr. Min., 1896, *19*, 81.
Pyramids: (883), ($\bar{3}.3.20$).
A. Krejčí. Ber. böhm. Ges. Wiss., 1898, *9*.
Pyramids: $N(\bar{2}0.39.60)$, $\sigma(141)$, $e(4.15.3)$, $g(1.24.2)$.
G. Boeris. Att. Soc. Milano, 1902, *41*, 357.
Dome: (201).
L. Hugo. Centralbl. Min., 1903, 121.
Pyramids: ($\bar{3}29$), (216), ($\bar{3}19$), (7.5.35).
F. Slavik. Zeitschr. f. Kryst., 1904, *39*, 301.
Pyramid: $R(113)$.

TOPAZ.

L. Sonheur. Zeitschr. f. Kryst., 1892, *20*, 231.
Prisms: (290), (580), (1.16.0)?, (14.15.0)?.
Pyramids: (415), (10.3.13), (8.7.15), (116)?, (11.2.13)?, (25.8.35)?, (437)?, (20.17.3)?, (459)?, (10.13.23)?, (20.27.47)?, (40.57.97)?, (10.31.41)?, (1.32.33)?.
P. Jeremejev. Zeitschr. f. Kryst., 1893, *22*, 74.
Pyramid: (338).
H. Bücking. Ber. Senckenberg Ges., 1896, 147.
Pyramids: (9.9.20), (1.11.12)?.
W. D. Matthew. S. of M. Quar., *14*, 53.
Prism: (4.11.0).

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G. Cesàro. Bull. Soc. fr. Min., 1889, *12*, 419.
Pyramid: (7.7.16).
F. Slavik. Bull. Acad. Sc. Böhm., 1902.
Dome: (107).
C. Anderson: Rec. Aust. Mus., 1908, *7*, 61.
Dome: *j*(501).

TOURMALINE.

G. d'Achiardi. Att. Soc. Tosc. Mem. Pisa, 1894, *13*, 229.
Trigonal Pyramid: (2.0.2.11).
Ditrigonal Pyramid: (5161).
G. Reimann. Neues Jahrb. f. Min. Beil.-B., 1907, *23*, 91.
Prisms: (10.9.19.0), (7.3.10.0), (3140), (10.3.13.0), (11.3.14.0), (7290), (11.2.13.0), (6170), (40.3.43.0), (15.1.16.0), (25.1.26.0), (30.1.31.0).
A. H. Westergård. Zeitschr. f .Kryst., 1906, *42*, 276.
Prism: (17.8.25.0) ?.
V. von Worobieff. Zeitschr. f. Kryst., 1900, *33*, 263.
Prisms: (10.7.17.0), (11.8.19.0), (5490).
Second Order Pyramids: (1129), (2.2.4.11), (5.5.10.14), (5.5.10.12), τ(2243).
Trigonal Pyramids: η(2025), (4047), (2023), (5054), (10.0.10.7), *i*(2021), (14.0.14.5), *F*(3031), *m*(13.0.13.4), (9092), (11.0.11.2), *P*(7071), (13.0.13.1), (0115), (0114), (0113), (0449), (0558), (0557), (0.13.13.14), (0.11.11.10), (0887), *N'*(0552), (0.11.11.4), ψ(0331), π'(0771), $\mathfrak{H}$(0881), (0.19.19.2).
Ditrigonal Pyramids: (16.1.17.15), (12.1.13.11), (10.1.11.9), ρ'(6175), $\mathfrak{f}$(4153), β(11.3.14.8), $\mathfrak{c}'$(10.3.13.7), (14.5.19.9), (17.7.24.10), (11.5.16.6), (13.7.20.6), (9.5.14.4), (7.4.11.3), *h*(5382), *y*(7.5.12.2), (15.11.26.4), *H'*(4371), (9.7.16.2), $\mathfrak{Z}$(13.12.25.1), *T*(18.17.35.1), *J'*(5491), (7.6.13.1), *L'*(15.13.28.2), $\mathfrak{ch}$(20.19.39.1), $\mathfrak{a}$(1.6.7.10), ϕ(4.11.15.14), (11.2.13.3), *D'*(7292), π(2132), $\mathfrak{R}$(2358), *Q*(1235), *L*(2.5.7.12), (11.2.13.12), (9.2.11.10), (7298), (9.4.13.11), (4265), *w*(3254), (3.8.11.7), (3.10.13.8), *M*(1453), $\mathfrak{e}$(3.14.17.10), (1895), $\mathfrak{g}$(1652), $\ddot{u}$(9.26.17.4), *z*(5.12.7.1), *D*(6.14.8.1), $\mathfrak{D}$(8.2.10.3), *X*(9.1.10.2), *K*(6281), (3.14.17.7), $\mathfrak{P}'$(3.10.

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$\bar{1}3.5)$, $\mathfrak{M}(2683)$, $B(1231)$, $T(3472)$, $(52\bar{7}1)$, $(15.4.\bar{1}9.2)$, $C(\bar{2}0.7.27.\bar{7})$, $j(1454)$, (1787), (2979).

TECHMANNITE.

Hexagonal. Tri-rhombohedral.

R. H. Solly. Min. Mag., 1905, *14*, 75.

Base: $o(0001)$.

Prisms: $a(11\bar{2}0)$, $b(10\bar{1}0)$, $d(31\bar{4}0)$, $f(71\bar{8}0)$.

First Order Rhombohedron: $r(10\bar{1}1)$.

Third Order Rhombohedrons: $x(13\bar{4}1)$, $z(24\bar{6}1)$.

G. F. Herbert Smith and G. T. Prior. Min. Mag., 1907, *14*, 300.

Prism: $D(25\bar{7}0)$.

First Order Rhombohedrons: $s(02\bar{2}1)$, $e(10\bar{1}2)$, $\alpha(40\bar{4}1)$.

Second Order Rhombohedron: $p(11\bar{2}3)$.

Third Order Rhombohedrons: $v(21\bar{3}1)$, $n(14\bar{5}3)$, $g(35\bar{8}4)$, $\zeta(35\bar{8}1)$ (other less important forms).

THÖGERITE.

Tetragonal(?).

V. Goldschmidt. Zeitschr. f. Kryst., 1899, *31*, 468.

Base: $o(001)$.

Prisms: $n(010)$, $\Sigma(120)$.

Second Order Pyramids: $y(012)$, $P(011)$, $h(032)$, $i(021)$.

First Order Pyramids: $t(111)$, $u(331)$.

TRONA.

Couyat. Bull. Soc. fr. Min., 1908, *31*, 343.

Dome: $(\bar{1}02)$.

URANOTIL.

Triclinic.

P. Pjatnitzky. Zeitschr. f. Kryst., 1892, *21*, 74.

Pinacoids: $A(100)$, $B(010)$, $C(001)$?.

Prisms: $l(210)$, $r(6\bar{1}\bar{0})$, $u(160)$?, $o(170)$?, $t_1(1.12.0)$?, $\omega(1.20.0)$?.

Domes: $T(011)$?, $D(101)$?.

Pyramids: $p(111)$?, $s(\bar{1}11)$?, $\pi(\bar{1}33)$?, $\Theta(\bar{1}33)$?, $\Sigma(11.20.11)$?, $\delta(\bar{2}0.9.9)$?, $\alpha(17.1.1)$?, $\mu(11.1.1)$?, $\beta(10.\bar{1}.1)$?, $\gamma(10.\bar{1}.1)$?, $\epsilon(8\bar{1}1)$?, $\sigma(7\bar{1}1)$?.

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URBANITE.

Monoclinic.

Hj. Sjögren. Geol. Fören. Förh., 1892, *14*, 251.

Pinacoids: $a(100)$, $b(010)$.

Prism: $m(110)$.

Domes: $p(\bar{1}01)$, $r(053)$.

Pyramids: $u(111)$, $n(221)$, $s(\bar{1}11)$, $o(\bar{2}21)$, $x(322)$ or (433)?, $y(\bar{6}14)$.

VALENTINITE.

A. Pelloux. Rend. Acc. Linc., 1900, *9*, (1), 340.

Pinacoid: $c(001)$.

Prism: $n(17.7.0)$.

Pyramid: $t(4.6.21)$.

VANADINITE.

V. Goldschmidt. Zeitschr. f. Kryst., 1900, *32*, 561.

Prisms: $g(31\bar{4}0)$, $f(51\bar{6}0)$.

First Order Pyramids: $\delta(50\bar{5}4)$, $\gamma(40\bar{4}3)$, $\alpha(30\bar{3}2)$, $\beta(50\bar{5}3)$, $\pi(40\bar{4}1)$.

Third Order Pyramids: $\xi(41\bar{5}4)$, $i(21\bar{3}2)$, $e(32\bar{5}3)$, $l(32\bar{5}2)$, $t(52\bar{7}2)$, $p(41\bar{5}2)$?.

VESUVIANITE.

J. Boecker. Zeitschr. f. Kryst., 1892, *20*, 225.

Dome: $T(106)$.

Pyramids: $S(229)$, $V(552)$, $W(14.14.5)$, $D(18.5.5)$, $K(722)$, $(13.13.4)$?.

H. Buttgenbach. Ann. soc. géol. Belg., 1898, *25*, 106.

Pyramids: (631).

M. Wiebull. Zeitschr. f. Kryst., 1896, *15*, 1.

Prisms: (310), (650).

VIVIANITE.

L. J. Spencer. Min. Mag., 1907, *14*, 324.

Prisms: $h(250)$, (130)?, (140)?.

Pyramid: (838)?.

WELLSITE.

Monoclinic.

J. H. Pratt and H. W. Foote. Am. Jour. Sci., 1897, *3*, 443.

Pinacoids: $a(100)$, $b(010)$, $c(001)$.

Prism: $m(110)$.

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Whewellite.

A. Weisbach. Neues Jahrb. f. Min., 1884, *2*, 48.
Prism: $l(130)$.
Domes: $k(102)$, $s(014)$.

F. Kolbeck and V. Goldschmidt. Centralbl. Min., 1908, 659.
Pinacoid: $a(100)$.
Prism: $v(250)$.
Domes: $d(032)$, $\alpha(031)$.
Pyramids: $g(111)$, $\gamma(122)$, $\epsilon(114)$, $\zeta(\bar{2}11)$, $\eta(211)$, $A(321)$, $B(341)$, $\beta(\bar{1}31)$, $C(\bar{6}51)$, $\delta(238)$.

Willemite.

S. L. Penfield. Am. Jour. Sci., 1894, *47*, 305.
Prism: $h(3\bar{1}\bar{2}0)$.
First Order Rhombohedron: $s(0111)$.
Second Order Rhombohedron: $s(1123)$.
Third Order Rhombohedrons: $u(2\bar{1}13)$, $v(1325)$, $x(3121)$.

O. B. Böggild. Min. Grönl., 1905, 276.
Prism: $(5\bar{2}70)$.

Wolframite.

C. H. Warren. Am. Jour. Sci., 1901, *11*, 372.
Prism: $i(7.11.0)$.

A. J. Moses. Am. Jour. Sci., 1905, *20*, 281.
Pyramid: $p(214)$.

O. B. Böggild. Min. Grönl., 1905, 180.
Prism: (610).

Wulfenite.

C. A. Ingersoll. Am. Jour. Sci., 1894, *48*, 193.
Second Order Pyramid: $p(201)$.
Third Order Pyramid: $\pi(313)$.

B. K. Emerson. U. S. Geol. Surv., 1895, Bull. *126*, 176.
Second Order Pyramids: $\theta(1.1.12)$, $\gamma(443)$.
Third Order Pyramid: $\lambda(131)$.

E. Artini. Riv. Min. Ital., 1896, *16*, 25.
Third Order Pyramid: (5.1.75)?.

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XANTHOCONITE.*

(Shown by Miers to be monoclinic.)

H. A. Miers. Min. Mag., 1893, *10*, 185.
Pinacoids: $a(100)$, $c(001)$.
Prism: $m(110)$.
Domes: $d(\bar{5}01)$, $D(501)$, $n(053)$.
Pyramids: $p(111)$, $q(551)$, $P(\bar{1}11)$, $Q(\bar{5}51)$, $r(112)$, $t(223)$, $h(334)$, $f(115)$, $y(443)$, $\rho(332)$, $R(\bar{1}12)$, $Y(\bar{4}43)$.

ZINCITE.

P. Grosser. Zeitschr. f. Kryst., 1892, *20*, 354.
First Order Pyramid: $o(20\bar{2}3)$.
A. J. Moses. S. of M. Quar., 1895, *16*, 226.
First Order Pyramids: $\alpha(40\bar{4}5)$?, $\beta(50\bar{5}4)$?.

ZINKENITE.

L. J. Spencer. Min. Mag., 1897, *11*, 188.
Pinacoids: $a(100)$, $c(001)$.
Dome: $\epsilon(102)$.

ZINNWALDITE.

G. Flink. Medd. om Grönl., 1898, *14*, 232.
Dome: (017).
Pyramids: (114), ($\bar{1}$.1.11), ($\bar{2}$21).

ZIRCON.

P. Jeremejev. Vh. Russ. Min. Ges., 1895, *33*, 429.
First Order Pyramids: $w(501)$, $u(701)$.
Ditetragonal Pyramids: $\rho(545)$, $r(766)$, $\pi(643)$.

* Also includes rittingerite.

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ABSTRACTS OF RECENT MINING DECISIONS.*

LOCATION OF CLAIMS.

Boundaries of Mining Claim—Field Notes and Monuments.

The courses and distances in the field notes and in the patent of a mining claim are not conclusive of the true locations of the established monuments of the official survey. But a competent witness may testify that he saw and observed the posts and monuments and knew that they were not changed from their original position.

Grand Central Mining Co. *v.* Mammoth Mining Co., (Utah) 104 Pacific 573, September, 1909.

Location of Mining Claim—Declaratory Statement.

The declaratory statement of the discovery of a lode mining claim sufficiently identifies it, if under any reasonable construction, in view of the surrounding circumstances, the description therein is sufficient to notify a subsequent locator that the particular part of the ground claimed has been located. On this theory a lode mining claim location was declared void, where the discovery shaft was made within the boundaries of a prior location. The laws of Montana do not require the declaratory statement of a discovery of a lode mining claim to describe the claim by metes and bounds; and where the original monuments have disappeared, the calls for distances in the declaratory statement will control over testimony as to the location of the original monuments, and a claim ascertained from bounds clearly fixed by such calls, such as discovery shafts, will prevail over a subsequent conflicting location, and any deviation from the calls in the declaratory statement filed upon the discovery of a lode claim in making a survey, would not prejudice an adverse claimant where such deviation reduced the length of the end lines of the claim where it conflicted with an adverse claim; as the lines of a lode claim could not be fixed as to include ground not covered by the original location. The fact that a subsequent location overlapped the west end line and included the discovery, would not affect the validity of a prior location, where such location was supported by a discovery shaft which was bisected by the west end line of a claim.

Tiggeman *v.* Mrzlak, (Mont.) 105 Pacific 77, November, 1909.

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* Prepared for the School of Mines Quarterly by J. W. Thompson, attorney-at-law, Indianapolis, Indiana.

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CONFLICTING QUARTZ LOCATIONS.

A quartz location, laid out substantially over a prior quartz location, gives no right to the claim in so far as the boundaries of the two locations conflict.

Strickland *v.* Commercial Mining Co., (Oregon) 104 Pacific 965, November, 1909.

OVERLAPPING CONFLICTING CLAIMS.

A discovery shaft which bisected the west end line of a mining claim was sufficient to support a lode location; and a subsequent location which overlapped the west end line and included the discovery would be void to the extent of the conflict. And in an action to determine the conflicting lode location, the burden was upon the last locator to prove by clear and convincing evidence that the first locator's claim was subject to forfeiture and relocation because of the failure to perform the required representation work.

Tiggeman *v.* Mrzlak, (Mont.) 105 Pacific 77, November, 1909.

LOCATION OF CLAIM—AMENDMENT OF CERTIFICATE.

If the certificate of the original location of a mining claim was insufficient because so defective as to prevent one from identifying or designating the claim on the ground, an amended certificate cannot be filed curing the defect where it includes other or different territory so as to injure intervening rights.

Washington Gold Mining and Milling Co. *v.* Olaughlin, (Colo.) 105 Pacific 1092, December, 1909.

PRIORITY OF LOCATION—FRACTIONS OF A DAY.

Where the relative priority of conflicting mining locations depends on the exact hour of the day of the filing of a location certificate, fractions of a day must be taken into account; and where a location certificate, not filed within the statutory time was filed in advance of the filing, on the same day, of a junior location certificate, in a controversy as to the priority, the former was held to be prior in time.

Washington Gold Mining and Milling Co. *v.* Olaughlin, (Colo.) 105 Pacific 1092, December, 1909.

LOCATION OF MINING CLAIM—RECORDING NOTICE.

The recorded notice of a mining claim need only be similar to that posted upon the grounds and show that the person claims the land described and identified therein; and where a notice recorded was a copy of a posted notice which was insufficient because it was not posted within the claim, and the fact that a valid notice was subsequently posted, which differed from the first notice only in omitting the name of a witness, but was not again recorded, was said to be immaterial as against subsequent claimants. In other words a valid notice of location may be posted after the notice is recorded.

Green *v.* Gavin, (Cal.) 105 Pacific 761, December, 1909.

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MINING CLAIM—ABANDONMENT—SECOND LOCATION.

Mineral ground covered by a valid location thereby becomes segregated from the public domain and is the property of the locator, and so long as he complies with the state and the United States Laws and local regulations he has the exclusive right and enjoyment of all the surface included within the lines of the location; and during such time the ground so segregated is not open to location by another. However grounds embraced in a mining location may become a part of the public domain so as to be subject to another location before expiration of the statutory period for performing the annual labor, if at the time of making the subsequent location there had been an actual abandonment of the claim by the first locator; but at the time of the making of such subsequent location the ground must be open to location under the mineral laws of the United States.

Swanson *v.* Kettler, (Idaho) 105 Pacific 1059, January, 1910.

LOCATION OF CLAIM—ABANDONMENT.

A senior locator, possessed of a paramount right in mineral lands for which a patent is sought, may abandon such right and thereby render the ground covered by such location subject to relocation before the expiration of the statute of limitations within which the annual labor must be performed. And upon application for a patent a junior locator, may show, as against the senior location, that the ground covered by such senior location had become a part of the public domain so as to be subject to relocation even before the expiration of the statutory period for performing the annual labor, by showing that there had been an actual abandonment of the claim by the first locator.

Swanson *v.* Kettler, (Idaho) 105 Pacific 1059, January, 1910.

STATE REGULATION OF MINING LOCATIONS.

A state may add to the regulations made by Congress for the location of mining claims; but such state regulations must not be inconsistent with those adopted by Congress. And a failure to comply with the state laws and rules will work a forfeiture whether such laws and rules provide for forfeiture for non-compliance or not, and the mining claim becomes subject to relocation by any qualified locator. Such statutes are said to be mandatory and substantial compliance with their provisions are necessary to perfect a valid location.

Knutson *v.* Fredlund, (Wash.) 106 Pacific 200, January, 1910.

LOCATION OF MINE—SUFFICIENCY OF NOTICE.

The statute of Washington requires a lode discoverer to record in the county auditor's office within a certain time a notice containing the name of the locator, the date of the location, the number of feet in length claimed on each side of the discovery, the general course of the lode, and such a description of the claim by reference to some permanent monument as will sufficiently identify it. In a controversy arising under a notice of a lode location under these statutes, it was

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held that location notices which did not designate the point of discovery, or state the number of feet in length claimed on each side of the location, or the general course of the lode, or identify the claim by reference to some natural monument, were insufficient; and in an action to recover possession of such a claim it was also held that an amended notice of location which did not cure the defects in the original notice was not proper evidence.

Knutson *v.* Fredlund, (Wash.) 106 Pacific 200, January, 1910.

CONFLICTING RIGHTS.

Conversion of Ore—Measure of Damages.

Where a person unintentionally and in the honest belief that he is exercising his lawful rights, entered upon the property of another, and removed ore, coal and timber, he was liable in damages for the value of such ore, coal and timber in their original place, and for no more. However if a person wilfully and intentionally, or with reckless disregard of the rights of another, takes his ore or timber and appropriates it to his own use, he must respond to the owner for the full value of the property at the time he converts it to his use, without deduction for the labor bestowed or expenses incurred in removing and preparing it for market; and such wrongful taking, in the absence of all other evidence, raises the presumption that such trespasser took the property intentionally and wilfully, or in reckless disregard of the rights of the owner.

Central Coal & Coke Co. *v.* Penny, 173 Federal 340, September, 1909.

Title to Mining Claim in Philippine Islands.

The possession and working of a mining claim in the Philippine Islands for the time required by the United States statute, in order to establish a right to a patent, need not have been under a claim of title.

Reavis *v.* Fianza, 30 Supreme Court Reporter 1, November, 1909.

Location of Mining Claim—Right to Patent.

A person who has initiated a right to a mining claim, recording his location notice, and performed the other acts made necessary by law, will be entitled to a patent on application, unless adverse rights are set up in the mode provided by the statute. And in an application for a patent by a junior locator, upon the failure of the senior locator to adverse, it will be presumed that there was no senior location, and that at the time the junior location was made the ground was open to entry under the mineral laws of the United States; but where it is made to appear that there is a third location made subsequent to such junior location, such third or subsequent locator may adverse the application for patent by the junior locator, and show that the junior location was void, because when made the ground was not open to location under the mineral laws.

Swanson *v.* Kettler, (Idaho) 105 Pacific 1059, January, 1910.

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MINING LEASES.

Construction of Oil Lease—Abandonment.

An oil lease gave the lessee the right for ten years to explore for oil and gas and provided that if a well was not completed on the premises within three months from the day of the lease the lessee should pay to the lessor a quarterly cash rental on advance for each additional three months the completion of any well was delayed. In an action on the lease it was held to be an executory contract and vested no title in the lessee to the oil and gas in place; that the lease contemplated the development of the premises within a reasonable time, and the lessee could lose his rights thereunder before the expiration of the period stated by abandonment of the lease, though it contained no forfeiture clause, and his intention to abandon the premises could be established by proof of such facts and circumstances as would show a voluntary waiver of his rights.

Smith *v.* Root, (W. Va.) 66 Southeastern 1005, January, 1910.

Construction of Gas Lease—Cancelation and Forfeiture.

An oil and gas lease, on a sufficient consideration, gave to the lessee the exclusive right to enter upon, operate for, and produce oil and gas from a certain tract of land situated in the gas belt. The lease provided that if a well was not drilled upon the premises within one year, the lessee's right should cease, and no alternative in payment of rent was given. The lessors were to have gas for domestic use, if found in quantities sufficient to justify the expense of marketing, and were to be paid $50 per year during the time gas should be marketed from each producing well. One well was completed within the year, yielding gas sufficient for marketing as provided in the lease. The lessors were given sufficient gas therefrom for domestic use, and the lessee also used it for fuel to furnish power in drilling two wells on other leases; but no other use was made of the well, and no gas was sold or marketed therefrom; no other wells were drilled, and the lessees never received anything except the one dollar, the consideration in the lease, and such gas for domestic use. Several other gas wells were drilled on adjacent lands by the lessee and by other parties, some of which yielded gas sufficient for marketing. More than four years after this well was completed the lessor served notice of forfeiture and commenced an action to cancel the lease. On the trial of the case it was held that the lease contemplated that the well should be operated and gas marketed therefrom in a reasonable time, and that other wells should be drilled and operated with reasonable diligence to utilize the lease; that four years delay under the circumstances was unreasonable; that the lessors had no adequate remedy in damages and were entitled to have the lease forfeited and cancelled.

Howerton *v.* Kansas Natural Gas Co., (Kans.) 106 Pacific 47, January, 1910.

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ASSESSMENT WORK.

Assessment Work on Mining Claim—Construction of Statute.

The statute of Washington providing that any mining district may make road building to mining claims within such claims applicable as assessment work upon mining claims, on certain conditions therein named, was held to apply only to organized mining districts. But independent of the statute work, done in connection with the claim, but outside of it, if done for furthering the development thereof, is available as assessment work as if done within the boundaries of the claim itself; and the building of a road, where there is no organized mining district, to be used in the general development of the mining property, is a doing of assessment work within the meaning of the law.

Sexton *v.* Washington Mining & Milling Co., (Wash.) 104 Pacific 614, October, 1909.

Abandonment of Claim—Relocation.

Where the appearance of a mining claim shows an abandonment of the premises for many years, and there are no boundaries marked by monuments, another location may be made on the same ground on the assumption that all possessory rights have been relinquished, and where the work done on such placer location was not sufficient to give notice of a valid possessory right in the claim as against subsequent locators.

Strickland *v.* Commercial Mining Co., (Oregon) 104 Pacific 965, November, 1909.

Location of Mining Claim—Effect of Failure to Work.

The statute of Montana requires performance of the following acts in order to constitute a valid quartz lode mining location: (1) Discovery of a vein or lode; (2) the posting of a notice of location at the point of discovery containing certain designated matters; (3) the marking of the boundaries on the ground and the doing of certain development work; (4) the filing for record of a declaratory statement containing the matters mentioned in the statute. Under this statute the failure, among other things, to perform the designated work will forfeit the location. But no one but the government or a subsequent locator can complain of a failure on the part of the original locator to do the necessary annual work, and a subsequent locator is not in a position to make complaint until he himself has completed a valid location; and if prior to the completion of such valid subsequent location, the original locator has resumed work upon his claim in good faith, he thereby saves his claim from such forfeiture and such subsequent locator can take no advantage of such delinquency. And in all actions or controversies for the forfeiture of mining claims the rule is that every reasonable doubt will be resolved in favor of the validity of a mining claim as against any effort or action of forfeiture.

Thornton *v.* Kaufman, (Mont.) 106 Pacific 361, January, 1910.

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LOCATION OF CLAIM—ASSESSMENT WORK.

Where a junior locator contended that annual assessment work was insufficient because the work was done for the purpose of representing several claims, a court, in determining the sufficiency of the work will apply the labor shown to have been done to the particular claims upon which the work was in fact done.

Swanson *v.* Kettler, (Idaho) 105 Pacific 1059, January, 1910.

FORFEITURE OF CLAIM—NONPERFORMANCE OF ASSESSMENT WORK.

The nonperformance of assessment work by the locator of a mine will not forfeit his claim thereto unless there has been a valid location by another, and a person who did not himself make a valid location cannot question the sufficiency of the assessment work done by the original locator.

Knutson *v.* Fredlund, (Wash.) 106 Pacific 200, January, 1910.

LIABILITY OF INJURY TO MINER.

INJURY TO MINER—PROXIMATE CAUSE.

A miner while riding out of the mine on the front bumpers of a cable car drawn up a slope, with his arm resting on the top edge of the car, came in contact with a stream of water escaping from a defective joint in a pipe used for pumping water from the mine, supported along the roof of the tunnel, and by reason thereof he was thrown around to one side and his arm was crushed against the roof of the tunnel, which was some six inches above the top of the car. In an action for the injury in order to render the mining company liable it was decided that it was not necessary for him to show that the force of the water was sufficient to physically throw him around and bring his arm in contact with the timber; but that it was sufficient if it caused him to instinctively throw up his arm, and thereby receive the injury.

Tremont Coal & Coke Co. *v.* Johnson, 172 Federal 785, September, 1909.

INJURY TO MINER—UNSAFE PLACE—ASSUMPTION OF RISK.

A miner has the right to assume that the mine owner has performed his duty to provide and maintain a reasonably safe place in which to work, and he is not required to exercise care to ascertain if the place is safe, where he does not himself create such place; and such miner assumes a risk from an unsafe place only when he knows it to be unsafe, or its dangerous condition is obvious. And in an action by a miner injured by the falling of a rock from the roof, it was held proper to admit the opinions of practical miners familiar with the place, to aid the jury to determine whether the roof was unsafe, and if so whether the parties should have known it in the exercise of reasonable care.

Central Coal & Coke Co. *v.* Williams, 173 Federal 337, October, 1909.

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INJURY TO MINER—CONTRIBUTORY NEGLIGENCE.

A miner purchased from his employer a fuse with which to set off a charge of dynamite in his employer's mine. On being properly lighted the fuse failed to fuse and smoke in the usual manner, and thereupon the miner supposing that it had gone out, frazzled the end of it and again applied the blaze, and on its failure to fuse again, repeated the operation, and thereupon an explosion immediately followed and the miner was injured. In an action by the miner for the injuries it was held that his complaint was insufficient, as he was not warranted in assuming that the fuse was defective because it failed to smoke, and that he was guilty of contributory negligence in remaining so near after first lighting it.

Cody *v.* Norton Coal Co., (Va.), 66 Southeastern 33, November, 1909.

OPERATION OF MINE—DUTY TO SUPPORT ROOF.

Under a statute providing that the owner or persons in charge of a mine shall keep a sufficient supply of timber to be used as props always convenient and ready for use, and shall send props down when required, and deliver them to the places where needed, it was decided that the duty to furnish the necessary props was imposed on the owners or operators of the mine, while the duty to use the props was imposed on the employes.

Lammey *v.* Center Coal Mining Co., (Iowa) 123 Northwestern 356, November, 1909.

DEATH OF MINER—LIABILITY FOR FAILURE TO FURNISH PROPS.

Where a miner and his son had sole charge of a particular part of a mine, and themselves created the dangers in the operation of it, the owner and operator of the mine was not liable for injuries which resulted in the death of such miner from a fall of the roof, unless its failure to furnish props on request was the proximate cause of the injury.

Lammey *v.* Center Coal Mining Co., (Iowa) 123 Northwestern 365, November, 1909.

INJURY TO MINER—NEGLIGENCE OF FELLOW SERVANT.

The owner and operator of a coal mine was not liable for injuries to a miner causing his death, where such injuries resulted from the negligence of a fellow servant.

Lammey *v.* Center Coal Mining Co., (Iowa) 123 Northwestern 365, November, 1909.

INJURY TO MINER—LIABILITY OF OWNER FOR NEGLIGENCE.

Where a miner suing for injuries had observed a fall of slate from the roof of a mine entry and called the attention of the pit boss to the condition of the roof, and such pit boss after examination found it dangerous and promised to make it safe and did cause a set of timbers

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to be put in near such place; and where it appeared that the mine had two expert timbermen whose duty it was to daily inspect the roofs and protect them by timbering, which fact was known to such miner; and that the entry containing such defective roof was continued to be used constantly by employes and miners after the first fall of slate, of which the miner had knowledge; and that such injured miner had not inspected the roof from the time of his first complaint to the pit boss, and it appearing that the duty of inspecting the roof was not mutual, and that the mine owner had or should have had knowledge of the facts which the injured miner did not possess and was not bound to discover, it was held in an action for injuries to the miner from the falling roof that the mine owner was negligent and liable for the injury.

Cotton *v.* Center Coal Mining Co., (Iowa) 123 Northwestern 381, November, 1909.

Injury to Miner—Imputed Negligence.

Where a father and son worked together in a particular room of the mine, and the supporting of the roof by props was left principally to the son; who was the father's helper, his failure to get props, or to use those which were furnished, and by reason of which the father was killed by the fall of a part of the roof, was negligence which was imputed to the father.

Lammey *v.* Center Coal Mining Co., (Iowa) 123 Northwestern 356, November, 1909.

Injury to Miner—Contributory Negligence.

Where two miners, working together in a mine, knew that the props necessary for supporting the roof had not been delivered, and that the roof as then propped was unsafe, and with such knowledge both continued to work therein without protest, when they were at liberty at any time to cease work and to determine their own hours of labor, over which the mine operator had no control, this was held to be such contributory negligence as would prevent one of them from recovering for injuries occasioned by the fall of the roof.

Lammey *v.* Center Coal Mining Co., (Iowa) 123 Northwestern 356, November, 1909.

Right to Sue for Injury Incurred in Mining.

Under the Pennsylvania statute giving a right of action to a person injured for failure to provide for the health and safety of persons employed in and about anthracite mines, and in case of death giving the widow and heirs the right to recover, it was decided that the word "widow" did not include a nonresident alien widow, and that a widow who was an Italian subject and was residing in Italy was not entitled to recover under the statute for the death of her husband in a coal mine in Pennsylvania, notwithstanding such a right might be enforced under the laws of Italy.

Debitulia *v.* Lehigh & Wilkes-Barre Coal Co., 174 Federal 886, December, 1909.

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Injury to Miner—Fellow Servant.

A mine boss and a fire boss employed in a coal mine, pursuant to the statute of West Virginia and in the performance of the duties thereby imposed upon them, including the duty of the mine boss to see that as the working places advanced "break-throughs" for air are made, or that "brattice" shall be used, were held to be fellow servants with a miner employed in such mine, and that the owner and operator of the mine was not liable for injuries sustained by the miner on account of the negligent performance of such duties by the mine boss.

Bralley *v.* Tidewater Coal & Coke Co., (W. Va.) 66 Southeastern 684, January, 1910.

Liability of Mine Owner for Injury to Servant of Independent Contractor.

The owner of a mine who employed an independent contractor to work the same, but retained control over the operation of the hoisting apparatus, was held liable for an injury to a servant of such independent contractor who was on the premises and in the position he occupied when injured under an implied invitation from the owner and for a purpose connected with the business which the owner was not only engaged in but which he permitted to be there carried on. While there was no privity between the owner and the servant, and no contractual duty from the owner to the servant, yet there was a mutuality of interest between the servant of the independent contractor and the owner, and the law imposes upon the latter the duty to such servant to use due care to employ a reasonably qualified hoister to handle the hoisting apparatus.

Sloss-Sheffield Steel & Iron Co. *v.* Bibb, (Ala.) 51 Southern 545, January, 1910.

COAL LANDS.

Validity of Entry of Coal Lands.

Under the United States Statute permitting the entry of coal lands by a qualified person, in case of an individual, not exceeding 100 acres, and in case of a corporation not exceeding 320 acres, persons cannot lawfully associate themselves together for the purpose of entering tracts of 160 acres each in severalty, but to be held for the joint benefit of all in equal shares, and patents so issued were cancelled at the suit of the United States.

United States *v.* Portland Coal & Coke Co., 173 Federal 566, October, 1909.

Conveyance of Coal Lands—Release of Damages from Work.

Where the grantors of coal under land released all claim for damages by the working of the mines in a proper manner, the removal of all the coal was not an improper working; and where a subsidence of the surface was caused thereby, the owner of the surface under such release had no right of action for damages.

Kellert *v.* Rochester & Pittsburg Coal & Iron Co., (Pa.) 74 Atlantic 789, October, 1909.

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Title to Coal—Ownership of Surface.

The ownership and possession of the surface land does not carry with it the possession of coal underneath such surface, where the estate in the coal had been previously severed as to title. And the surface owner in order to claim proper possession of coal severed in title from the surface land, must show that he has had actual physical possession of the coal, apart from his possession of the surface, such as by operating mines.

Plant *v.* Humphries, (W. Va.) 66 Southeastern 94, November, 1909.

Fraudulent Entry on Coal Lands.

By a fraudulent entry on coal lands a claimant acquired no rights therein, and his alleged title was void. Neither could he acquire any title to a coal mine thereon pending cancelation of such fraudulent entry, and his obtaining an injunction on the strength of his supposed right before cancelation of his entry was said to be a fraud on the jurisdiction of the court.

Baldwin Star Coal Co. *v.* Quinn, (Colo.) 105 Pacific 1101, December, 1909.

MISCELLANEOUS.

Option Purchase of Mine—Fraud.

A purchaser of mining property cannot defend in an action for the purchase price on the ground of fraud and deceit, where he admits he had heard rumors of claims of others to the property, but made no inquiry or effort to learn the truth of such rumors, except from the vendor.

Moore *v.* Pooley, (Idaho) 104 Pacific 898, October, 1909.

Laborer's Lien on Mines.

The statute of Oregon providing that "every person who shall perform labor," about a mine, shall have a lien on the property, was held to apply to ordinary laborers who perform actual visible toil with their hands, other kinds of service not being included, and did not include superintendents or managers.

Durkheimer *v.* Copperopolis Copper Co., (Oregon) 104 Pacific 895, November, 1909.

Mining Partnership—Authority of Partners.

Where the members of a mining partnership were not able to agree, it was held that those having the majority interest had the right to control the management in all things necessary and proper for the operation of the mine, and they could not be held liable in an accounting except for culpable negligence, breach of duty, wrongful conduct or diversion of the property from the business of the firm. And in an action for dissolution, accounting and winding up of the mining partnership, an inferior court was not authorized to enter a personal decree against one partner in favor of another for the balance found

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due him on a partial settlement. The social property should first be reduced to money and the proceeds applied to the discharge of the partnership liabilities, including any balance found due on final settlement from one partner to another, and then a decree over on final settlement for any balance that might remain. And in such a settlement one partner who had paid the share or interest of another in the operating expenses would be entitled to interest thereon, as against the delinquent partner, on dissolution and final settlement and winding up of the partnership.

Bartlett & Stancliff *v.* Boyles, (W. Va.) 66 Southeastern 474, November, 1909.

Devise of Coal Lands.

A testator by his will devised to one person the surface of certain lands described, and gave to another the coal under such lands. On a construction of the will in a controversy between the devisee of the lands and the devisee of the coal, it was held that the will gave to the former the land in fee simple, except the coal under it, and that it gave to the latter the coal, with the right to mine it, including the necessary roadway and the use of the surface necessary to the proper operation of the mine.

Myher *v.* Myher, (Mo.) 123 Southwestern 806, December, 1909.

Dower in Mining Property.

Coal in place is said to be "real estate" within the meaning of a statute giving a wife dower in all her husband's real estate, of which he was seised in fee simple at any time during the coverture.

Reynolds *v.* Whitecarver, (W. Va.) 66 Southeastern 518, December, 1909.

Mining—Liens for Labor and Material.

The procurement of the carbonaceous mineral commonly known as mineral coal, by digging in the earth is termed "mining"; and the term "mine" or "deposit yielding metals or minerals of any kind," used in the statute of Oregon giving laborers and material-men a lien for the working or development of any mine, lode, mining claim or deposit yielding metals or mineral of any kind, has been held to include a coal mine, although the term "mining claim" as used in such statute may not do so.

Escott *v.* Crescent Coal & Navigation Co., (Ore.) 106 Pacific 452, January, 1910.

Contract for Sale and Purchase of Coal.

A written proposition from the owner of a coal mine to the president of a railroad company suggesting the building by it of an eight mile branch line from its road to his coal mines, he to receive a guarantee of tonage of freight that would be satisfactory to the company, and an answer from the president of the company to the effect that the company would build the line for any parties who would furnish the right of way and put in a coal plant of not less than thousand tons

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of coal per day, and would furnish the coal at the same price as that furnished by the Pocahontas people, and that the company would take from such persons at that price whatever amount of coal they would agree to furnish, not less than one hundred thousand tons a year, or if they preferred to ship it themselves, the company would give them the same rate made to other parties. Thereupon the offer was accepted by the owner of the coal lands who stated that he would commence immediately to make leases and push his part of the contract with all promptness. And leases were accordingly made, the mines developed, the right of way for the road was furnished and the branch road was built. The mines were operated by lessees who sold their coal to the company, but after a number of years it refused to receive any more of the coal. In an action to enforce the agreement it was held by the Federal Court that the correspondence created a contract for purchasing the coal, and was not merely a proposal on the part of the railroad company to negotiate future contracts, and that such contract construed in the light of the situation of the parties and the language of the letters, did not limit the company to take such coal as the land owner personally should mine, but included all coal that should be mined by lessees of such mine owner.

Mckell *v.* Chesapeake &c. R. Co., 175 Federal 321, January, 1910.

Construction of Statute Giving Lien for Labor and Materials on Mines.

The title of the act of the Legislature of the State of Oregon giving laborers and material-men a lien for the working or development of any mine, lode, mining claim, or deposit yielding metals, or mineral of any kind, reads as follows: "An act for securing liens for laborers on mining claims and material-men and prescribing the manner of their enforcement." The constitution of the state of Oregon provides that every act shall embrace but one subject which shall be expressed in the title. In an action involving the rights of laborers and material-men to enforce a lien on a coal mine for work done and material furnished, and which involved a construction of this statute, it was held that the statute was not unconstitutional on the ground that it was broader than the title, in that the title purported to secure liens on "mining claims" only, and not to include mines generally. The court conceded that the term "mining claim" as used in the title was not clear, but the court would not resort to a critical construction of the title in order to hold a statute unconstitutional, but would give the language a liberal inerpretation, and the largest scope would be accorded to the words used that reason would permit, in order to bring within the purview of the title all the provisions of the act. And in constructing the statute the court would look to the purpose for which it was enacted, and the rule is that statutes providing for liens are to be construed in favor of the purpose of the law; and as the purpose of this statute was to protect laborers and material-men of a particular class, and the Legislature used the words "mining claim" in the title, such words would not be strictly constructed in determining the question of the

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right to a lien, but they are to control only in the matter of measuring the extent of the lien.

Escott *v.* Crescent Coal & Navigation Co., (Ore.) 106 Pacific 452, January, 1910.

DUMP FROM COAL MINES—DAMAGES FOR NUISANCE.

The dirt dumped from a coal mine making a permanent pile from 250 to 300 feet long, 100 feet wide at the base and 50 to 60 feet high, was held to be a permanent nuisance, and that only one action could be maintained for the recovery of damages, and that the measure of damages was the decrease in the value of the property injured thereby.

Risher *v.* Acken Coal Co., (Iowa) 124 Northwestern 764, February, 1910.

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UNIVERSITY NOTES.

According to an article in the September issue of the *American Journal of Science,* a new mercury mineral from Terlingua, Texas, has been named in honor of Professor Moses, of the Department of Mineralogy, who first definitely described the interesting mercury minerals found in Texas. The three minerals, montroydite, terlinguaite and eglestonite, were named and definitely determined by him, and he also gave a preliminary notice of a fourth new one (later named kleinite). It was in honor of this work that the discovers of the new mineral, F. A. Canfield, W. F. Hillebrand and W. T. Schaller, named the compound in question Mosesite, which name, beside perpetuating the high attainments of Professor Moses in the science of mineralogy, is particularly appropriate as it links his name with a group of minerals which he definitely put on record.

The new mineral, Mosesite, occurs in the form of small yellow crystals over crystals of calcite. A chemical analysis of the substance shows it is to be composed of a mercury-ammonium compound containing chlorine and the sulphate group, with a small percentage of water. The crystals have peculiar optical properties.

Professor C. E. Lucke is delivering a course of lectures on "Power" for the Department of Education of the City of New York. The object of this series of lectures is to point out the enormous effect that the substitution of mechanical power for hand and animal labor has had on the organization of society and the conditions of living, and by presenting the development of power machinery to show what sort of ideas have produced this result. The effect of power machinery is soon told and readily understood; but it is not so easy, though far more important, to show how the men who are responsible for what has been done have thought and worked, and with what kind of things and reasoning those men must deal who are to take up the responsibility for future progress. Accordingly, the bulk of the subject matter is concerned with the apparatus and machinery for the converting of natural energy in any of its available forms into useful work, together with the physical processes for the execution of which that apparatus was devised.

C. M. Fulton, School of Mines 1897, president and professor of metallurgy, South Dakota School of Mines, has published a most excellent volume on the "Principles of Metallurgy," which is reviewed in this issue.

A. L. Colby, School of Mines 1881, has written a book on "Reinforced Concrete in Europe," published by the Chemical Publishing Co., Easton, Pa.

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DEPARTMENT OF GEOLOGY.

(From the annual report of the head of the department.)

The year has passed quite satisfactorily and on the whole the work done in the department has been good. We plan one or two minor changes for next year which will add to efficiency. I will briefly review the courses.

Geology 4—Microscopic work with thin sections of rocks, brought us 59 students from the third-year class for the last two months of the second semester. Dr. Berkey took them in two sections, but even then we were taxed to the limit for microscopes. We had to make the best of some rather poor ones, but we hope that before another year some promised aid will help us out. The class worked admirably, often individuals remaining overtime. A very excellent impression for earnestness was left with us.

Geology 6—General geology for second-year School of Mines students. This is our regular introductory work, given six times weekly. We had 70 in the class. I do not like six times a week as well as three, as a method of instruction and coming in the second term, it very unevenly divided my work. Happily at this writing Professor Moses and I have arranged to alternate days in mineralogy and geology all through next year. The class was rather more restless than usual, and with three hours weekly I hope this feature will pass away.

Geology 16—Index fossils, or the fossils used in identifying strata. Formerly with the geological option this course found a constituency, but with the abolition of this privilege a year or two ago it became possible for only one or two fourth-year students, who were ahead of their schedule, to take it. At present the class is almost entirely limited to graduate students.

Geology 18—General Geology for second-year civil engineers. The work was given to forty students, and consisted of three lectures and an afternoon in the field. Dr. Berkey gives the instruction and utilizes to the full the exceptional advantages given by New York's great engineering enterprises for illustration and instruction. Cross-sections are drawn and not infrequently excellent maps are made. Much practice is given in the interpretation of topographical maps. In fact, quite careful efforts have been made to develop a line of treatment for civil engineers that will meet their needs and appeal to their good judgment. Dr. Berkey has done this very successfully.

Geology 19—Several years ago a number of the civil engineering students who had had Course 18, wished to take more. The same was true of some graduates, so that Dr. Berkey developed this course, which is strongly physiographic and in which a number of engineering problems, taken from life, are discussed. It is much the same for civil engineers as my course 110 for mining engineers. The actual cases of the use of geology excite much interest and the course has proved very profitable to from 10–15 students yearly.

Geology 20—Rocks and Soils. This is a new course especially

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started for students in forestry and agriculture. It comes twice weekly and has had 13 hearers. Dr. Berkey gives it and has shaped it for the constituency involved. It is a bit early for mature judgment as yet.

Geology 105–106—Economic geology, was given three times weekly by myself throughout the year. An extra hour was devoted to a conference on Saturdays, in which I go over the literature of the subjects discussed during the week and lay out a course of reading. The extra hour can only be taken by those mining engineers who are a little ahead of their required work. There were 75 in the regular class and from 5–15 at the extra hour. The class was a very attentive one and did good work. We are finely equipped for this course, but are yearly placing the museum in better shape. It will shortly be one great study collection, which I hope will be much used by those specially interested in the subject.

Geology 110—Geological Problems and Surveys. I have been at much pains to bring together a rather large series of cases of the applications of geology in mining or other engineering enterprises. For much of the term twice a week I go over them with the fourth-year mining engineers. I also assign a problem each week to some member of the class for solution and presentation. Toward the close of the term I give a brief history of national and state geological surveys, with the experiences of official geologists in keeping their organizations alive and in accomplishing their work. Not a few of our graduates have found the course suggestive and of subsequent service. Indeed, when the connection of our sciences with life is made clear, they never lack interest.

Geology 112—Field-work and Report. The week following the final examinations is given to this course which is taken by third-year mining engineers, a few college students and graduate students. A suitable quadrangle of the U. S. Geological Survey is selected and its topographic map is used as a base. With a party the size of our recent classes, we have been able to prepare a map of the entire area, and the results will answer for a first and fairly close approximation. In 1909 we spent a week at Pottsville, Pa., with a party of 65 students and 5 officers. The days are devoted to mapping one or two square miles which are assigned the night before to squads of two men each. The evenings are given over to reports and quizzes on the day's work. At Pottsville we had a section from the top of the Lower Silurian, through the Upper Silurian, Devonian, Carboniferous, and into the Devonian again. It was a rare exhibition of folded rocks, especially in the Schuylkill anthracite basin, and none of us will ever forget its exposures. The people of Pottsville did everything they could for us, and the students conducted themselves in a way to make the officers very proud of their party. Mr. Baird Halberstock, of the former Pennsylvania Geological Survey, was especially good to us. The trip gave the data for a report, geological maps and sections.

Geology 201–202—Petrology, or our long course in the microscopic study of rocks. I give the two lectures weekly and Dr. Berkey over-

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sees the laboratory work. Fourth-year mining engineers who are a little ahead of their course and graduate students make up the 10 to 20 who go over the subject with us yearly. The matter of the lectures is shaped so as to bear closely on the subject of ore deposits. The work has proved of interest and has been especially serviceable to those School of Mines men who afterwards have found their work in the examination of properties.

In general, in the department we three senior officers have shared the elementary courses, I taking the School of Mines students, Professor Grabau the College and Dr. Berkey the civil engineers. We each then follow out a special line, economic geology being mine, paleontology Professor Grabau's, and petrography Dr. Berkey's. Dr. Berkey and I are in closest touch with the engineers, and we endeavor to keep our work, as several of the courses have shown, in intimate connection with the experiences which inevitably come along in the life of an engineer.

J. F. K.

DEPARTMENT OF METALLURGY.

The annual report for year 1909–10 from the Department of Metallurgy to the dean was as follows:

Organization.—Professor Campbell was promoted from the rank of adjunct professor to that of associate professor. No regular assistant was appointed for the department, but Mr. Ralph H. Sherry acted in the capacity of honorory assistant during the year. There were no other changes in the organization.

Instruction.—A number of changes were made in the courses given, mainly in their rearrangement, the object aimed at being to give instruction in metallurgy in the proper sequence. In all, nineteen courses were offered and students were registered in all of them.

The total number of students for one or more of these courses was 264 as follows:

Special students	4
School of chemistry	30
School of engineering	131
School of mines	82
Candidates for advanced degrees	17
Total	264

Teaching.—At times during the past year the rush of work in this department has been so great that it could only be handled with difficulty by the present staff. If the work in the department continues to increase as apparently it will, it will be necessary to provide for a regular assistant in the near future.

The comparatively large amount of experimental and research work which is being done in the department occupies a great deal of the instructors' time and there is no means of expressing the amount of time thus occupied in terms of students' hours.

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Research.—Some very valuable research work has been done during the past twelve months, especially in determining the volatility of gold in alloys in a molten condition; the decomposition of metallic sulphates; the effect of impurities in electrolytes on metals deposited from solution; annealing of steel; heat treatment of manganese steel; the ternary alloys of the lead-tin-antimony series; the constitution of certain opaque ores, etc. This research work will form the basis of papers to be published in the future.

Summer Work.—During the months of June and July, 1909, an extensive trip was made by the summer school in metallurgy under the personal direction of Professors Walker and Campbell and Dr. Kern. The trip lasted six weeks and during this time the electrolytic copper refineries, lead refineries, smelting plants and copper converting plants in the vicinity of New York were visited. The class then left for the West, stopping at and inspecting the operations of various metallurgical plants which can be briefly mentioned as follows:

The New Jersey Zinc Company's Plant and the Bethlehem Steel Works, at Bethlehem, Pa.; the Pennsylvania Steel Company at Harrisburg, Pa.; the Cambria Steel Company, at Johnstown, Pa., and a number of plants in the vicinity of Pittsburg, where the class remained two weeks. At Chicago, the lead refining plant of the American Smelting and Refining Co., the electrolytic lead refinery of the U. S. Metals Refining Co., at Graselli, Ind., and the immense plant of the U. S. Steel Corporation at Gary were visited in turn. In Colorado various lead smelting plants, gold chlorination plants and cyaniding plants were seen and at Salt Lake City the operations of copper mining, smelting and converting were carefully studied.

The trip was quite an extensive one and gave the men an excellent idea of the various metallurgical operations which are involved in the extraction of metals from their ores and what was equally important, an idea of the breadth and scope of the Metallurgical industry. It will not be possible to make such a long trip this year and the summer metallurgical work will be confined to the vicinity of New York and Pennsylvania—the western trip being repeated the year after (1911).

During the past twelve months Professor Walker has made three professional trips; one embracing work in Chicago, Colorado and Oklahoma; the second to North Carolina and the third to the central portion of Arizona.

Professor Howe has been elected president of the International Association for Testing Materials and was also appointed chairman of the Committee on Alloy Steels of the American Society for Testing Materials of which Professor Campbell is the Secretary.

Professor Campbell has been elected a vice-president of the New York Academy of Sciences and has been appointed chairman of the Committee on Non-Ferrous Metals and Alloys for the American Society for Testing Materials. He has also been elected a member of the Iron and Steel Institute of Great Britain.

A. L. W.

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DEPARTMENT OF MINERALOGY.

(From the annual report of the head of the department.)

The department this year is giving instruction to 260 students and in 1,084 student hours.* This easily makes it the largest department of mineralogy in the country.

The naming of a newly discovered mercury mineral "Mosesite" after the head of the department by Messrs. Canfield Hillebrand and Schaller is recorded in the *American Journal of Science,* Vol. 30, p. 202.

At the request of the professors of geology and mineralogy the plan which has been tried for two years of concentrating each subject within one term has been abandoned, and hereafter, by an adjustment of the order of the lectures in each subject, the two courses will be given throughout the second year.

An undergraduate course in mineralogy about equivalent to the course given in Columbia to chemists and chemical engineers has been started in Barnard College. The necessary specimens and models are for the present provided by the Department of Mineralogy.

A request has been made to the Committee on Instruction to assign one lecture hour per week to mineralogy 15–16 given to chemists. The number of students has grown from 10 to 36 and the limiting of instruction to laboratory periods is no longer satisfactory.

A special lecture room accommodating 80 to 90 men is needed as the method of instruction followed requires that during the lectures specimens shall be before each student. Narrow tables are used and the usual "chair and arm" lecture room are not suited to the work.

A room for microscopic work with north light is badly needed. Instruction is given in several of the courses in the microscopic study of minerals and crystals. For this work the windows of the museum are fitted with tables, as in one course approximately 50 men are at work for six weeks and in other courses about 75 men do smaller amounts of the same work.

The *American Journal of Science,* October, 1910, contains an article by Professor Moses on the "Synthetic Sapphires of Verneuil" and the July number of the SCHOOL OF MINES QUARTERLY contains a "Guide to the Sight Recognition of Seventy Important Minerals."

The graduates in the field could relieve the department of a difficulty if they would from time to time ship 50 to 100 pounds of those minerals which occur in their locality in fair purity.

The large classes use up and destroy in their testing many hundreds of samples each year, and the replacing of these in the proper variety is difficult, for the dealers cater principally to collectors who have no use for ordinary specimens. We need especially common minerals of fair purity, minerals of economic value and minerals which are common alteration products.

A. J. M.

* That is students × hours per week for the year.

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DEPARTMENT OF MINING.

(From the annual report of the head of the department.)

During the past year Professor Munroe has been secretary of the Mining and Metallurgical Society of America and has attended the monthly meetings of the New York Section of this society, and taken active part in the discussions and committee work of this organization.

During the year he has also held the office of president of the Society for the Promotion of Engineering Education, and as the chairman of Programme Committee has had charge of the arrangements for the annual meeting held in June at Madison, Wis. He presided at that meeting and delivered an address on "Efficiency in Engineering Education," which served as an introduction to a symposium on that subject. He also organized a second symposium on the teaching of calculus. In order to bring about a full discussion of these important topics a large amount of correspondence was necessary by circular letters and by individual communications. These symposia proved most interesting and brought out a mass of material certain to prove of great value and likely to bring about important advances in engineering education throughout the country.

During the year Professor Munroe assumed the entire charge of graduate students in mining and conducted two conference classes in advanced work.

Professor Peele has been engaged during the year on a revision of his "Compressed Air for Mines" and has added much new and valuable material to that book. Professor Peele is now absent on leave of absence for the first half year. A letter was recently received from him dated at Cape Town, South Africa. He spent nearly a week at Kimberley, where he found Dickinson, 1900, in charge of the five big diamond mines of the De Beers Co., having under his charge 2,400 white men and from 14,500 to 16,000 natives. Professor Peele next went to Johannesburg where he spent two weeks visiting the gold mines. He found here a number of School of Mines graduates in charge of some of the most important operations in the district, including S. C. Thompson, 1893; B. P. Carter, 1895; A. M. Robeson, 1899; Ralph Martin, 1904; A. W. Evans, 1906; J. B. Overbeek and Victor Hartog, 1908.

In June the regular summer class in underground surveying with 51 students was held at the Orchard mine and at the Teabo shaft of the Wharton Steel Co., at Wharton, N. J. In this work Professor Munroe had the assistance of Professor Kurtz and Instructor Judd, with a group of six temporary assistants, M. C. Browne, H. L. Carr, J. I. Kane, J. Maller, C. F. Schnepp and F. A. Wardlaw.

As was the case last year the limited number of instruments made it necessary to put four and sometimes five men in a squad and to divide the class into three sections, each working eight hours per day. At the Orchard mine one section started work at seven o'clock in the morning and continued until three in the afternoon; the second section

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began work at three o'clock and continued until eleven o'clock at night, while the third section worked from eleven o'clock until seven the next morning. Each squad spent one day in shaft surveying at the Teabo shaft.

On the completion of this work the class was divided into small sections of from two to four men and assigned to various mines throughout the country for the study of mine plant and underground work. For this work students visited mines in Ohio, West Virginia, Tennessee, Michigan, Missouri, Colorado, Utah, Wyoming, Montana, California, British Columbia and Mexico.

During the year a photographic laboratory has been equipped in the attic of the School of Mines building and an expert photographer was engaged, the expense being divided between the several departments, A large amount of work was accomplished, including the making of negatives, lantern slides, blue prints and bromide enlargements. By this arrangement it was possible to secure work of the highest quality and at moderate cost. The bromide enlargements have proved a most satisfactory method of illustrating lectures and they have been made at far less cost than has heretofore been possible where the work has been done by outside parties. It is hoped that it will prove practicable to continue this experiment another year by securing the coöperation of other departments of the university.

H. S. M.

(Reports of other departments will appear in next issue.)

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BOOK REVIEWS.

A Manual for Assayers and Chemists. By W. H. SEAMON, B.S.A. New York, John Wiley and Sons. Large 12mo, xi + 255 pp. $2.50 net.

This book, to all intents, represents the practise in the author's laboratory, which, like most analytical laboratories in the western states, deals with ores, fluxes, fire essays for gold and silver, etc. The preface states that it was written "principally for the young graduate of the mining school, thrown upon his own resources for the first time," etc.

The text is in three parts: I., Metallic Determinations (arranged alphabetically); II., Non-metallic Determinations (includes Cl, P, Se and Fe (together), SiO_2 and S); III., Miscellaneous—dealing with cements, fuel, slag, cyanide practise, etc., which are followed by chapters of advice and suggestion, forms of ore contracts in Mexico, information for ore shippers, etc. Tables are appended, chiefly comparisons of weights and measures. The indication is that the book is more particularly intended for the use of analysts in Mexico.

Most of the methods recommended are good, though by many they would not always be considered the best for their own practise. They rather represent the individual preferences of the author, than what would be the choice of analysts in general. A few, such as the acidimetric titration of alumina, or the chromate titration of lead, are rejected as unsatisfactory by most analysts. With the exception of the volumetric method for mercury, none (except as to unimportant modifications) are original.

As regards the chemical reactions of the various processes, and comments on and reasons for the methods of manipulation, the book is very uneven, giving such points in a few cases with great fulness, but in too many instances such are omitted. References to the literature are very infrequent, and are almost entirely to papers which have appeared in the *Western Chemist and Metallurgist,* or to books by western authors. The fact suggests the conditions frequently obtaining in our western technical schools, viz., difficulty of access to current chemical literature in general. It may be that the author considers that only the processes described by western technical men are adapted to the conditions of that section.

The general impression produced is that the book is one-sided, and fails in many cases to indicate even the existence of methods other than those described, which methods are well known to most analysts, and are in common use by many. For instance, the idea conveyed by the chapter on Zinc is that there is no accepted method for its estimation (when over about 10 per cent.) other than the ferrocyanide titration. The use of the phosphate method for either zinc or manganese would seem to be unknown to the author. No suggestion is made

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as to which of the methods given are so generally accepted as to be regarded as standard, and which are not.

Such points as these seriously detract from the usefulness of the work to "the young graduate of the mining school," etc., for which it is said to be intended.

In the chapter on Platinum the author complains that he has not been given due credit in the papers of his colleagues for his description of the potassium iodide test which he published in 1893. He seems to be unaware that the test was well known long before that time, and that it is used, and has been used by many chemists who never knew of his existence. His explanation of the reaction—as being due to free chlorine—is altogether incorrect.

In the chapter of advice as to systematizing one's laboratory work so as to obtain the most results when under press of work, he cites an incredible case of an analyst and assayer who achieved 278 determinations (fifty of them fire assays for gold and silver, the rest wet analyses) in a working day of ten hours.

The book has many good features, but as a complete or satisfactory laboratory guide for those for whom it is intended, it is in many respects deficient. ELWYN WALLER.

Suggested Rules for Recovering Coal Mines after Explosions and Fires. By W. E. GARFORTH, Mem. Inst. C.E., F.G.S., Past President of the Mining Association of Great Britain, etc. New York, D. Van Nostrand Co., 1910. 12mo, 71 pp., with maps. Price $1.50.

The author, a past president of the Mining Association of Great Britain, has gathered into this small pocket-sized volume the fruits of his experience in the extinguishing of mine fires and the rescue of miners after an explosion. In order to condense the information into a small space for quick reference, a didactic style has been adopted, the several rules being plainly stated without discussion of the whys and wherefores. In the latter part of the volume, however, the case of an actual mine disaster is cited at length, with marginal references to the numbered rules in the earlier part of the volume, as indicating the value and appropriateness of the suggestions.

The rules are grouped into two sections: First those relating to the precautions that should be observed and the supplies provided *before* an accident occurs; and second, rules for guidance after an accident. Many of the rules of the first section are tacitly agreed to by all mine managers, and yet they are seldom observed. Such, for example, are the provision of accurate, clear and up-to-date mine maps and sections for the guidance of rescue parties, the maintenance of rescue stations and the training of a special squad of men for rescue service. The author is particularly insistent on the necessity of training men to the use of oxygen helmets under conditions approaching those encountered in a burning mine, believing that for an untrained man to put on one of these devices and go into a wrecked mine is no less than suicidal.

The rules in the second section advocate extreme caution and freedom from haste and excitement in dealing with the difficult problem

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of restoring ventilation and rescuing survivors. From a casual reading of the rules, it might appear that the rescuers would be so weighted down with the suggested paraphernalia and so busily engaged in entering their movements and intentions in special books, as to be hampered in their efforts, but the author quite properly remarks that "if one person is overcome by afterdamp, remember he is not only thrown out of action, but others are required to carry him back to safety, which may necessitate the return of the party." In other words, a blind bravery is less efficient in this particular kind of work than a cautious fortitude.

The little volume, the body of which can be read in forty-five minutes, is strongly to be recommended, not as a guide to be blindly followed under the excitement of the period just following the news of a disaster, but as a body of plain and carefully digested rules to be mastered beforehand and held in constant readiness for instant use.

E. K. J.

Principles of Metallurgy. C. H. FULTON, President and Professor of Metallurgy, South Dakota School of Mines. New York, McGraw-Hill Book Co., 1910. Pp. 528, index, 6 × 9. Illustrated, $5.00 net.

The book is divided into fourteen chapters as follows: I., The Metals and Metallurgy from the Historic Point of View; II., Physical Mixtures and Thermal Analysis; III., Physical Properties of Metals; IV., Alloys; V., The Physical Properties of Alloys; VI., The Measurement of High Temperature; VII., Typical Metallurgical Operations; VIII., Slags; IX., Matte, Bullion, and Speise; X., Refractory Material for Furnaces; XI., Fuels; XII., Combustion; XIII., Furnaces, the Production and Heating of Blast; XIV., An Example Illustrating the Physics and Chemistry of a Smelting Operation. The Blast Furnace for Copper.

Former text-books covering this subject have been either too elementary to be suitable for anyone but a beginner or too fragmentary to form a logical sequence, due no doubt to the fact that the old ideas of metallurgy were being overshadowed by the modern from the view point of physical chemistry.

The present volume is strictly modern, for, after a few pages of history as an introduction, it starts right off with a chapter in which the essence of metallography is given and the types of freezing-point curves classified, with examples. The next chapter takes up the Physical Properties of Metals and Alloys with a clearness and order hitherto unattempted in any text-book on the introduction to metallurgy. The sixth chapter gives briefly but thoroughly, an outline of modern pyrometry. Under Typical Metallurgical Operations, metallurgists will find much that is new and interesting, while the chapters on Slags, Matte, Bullion and Speise will, to many, read like heresy or reform. They show clearly that these metallurgical products are solutions and act as solutions might be expected to and are, therefore, normal and simple in their behavior and no longer things of mystery. The remaining chapters are equally clear and well written without any signs of padding.

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Taken as a whole the book fills a long felt want, but how great that want really was we do not realize till we have read it through. Taken in parts (or chapters), the book is a series of essays giving us in a wonderfully clear way what is now known of the subject given in the heading. For the student of metallurgy, this book will become a necessity, while for the practical man who has been too busy in the field to follow the more scientific periodicals, it will come as a relief for he can catch up those years of neglected reading so easily. The few mistakes are clearly misprints and oversights.

Printing, paper, illustrations, etc., in the McGraw-Hill standard of excellence.

We congratulate Mr. Fulton on giving us something we really need and giving us satisfaction.

W. C.

Dynamo Electric Machinery, Vol. I., Direct Current Machines. By S. SHELDON AND E. HAUSMAN, New York, D. Van Nostrand Co., 1910. 8th edition., 8vo, 310 pp. Illustrated. $2.50.

The latest edition of this very valuable text-book will undoubtedly meet with the success which has been awarded to the seven previous editions.

Nearly all of the subject matter of the present edition has been rewritten; matter which seemed out of place in the previous editions has been removed, while such subjects as commutation, predetermination of machine characteristics, etc., has been more fully treated than before. The excellent set of cuts, which serve to illustrate the text, are well chosen and are a delight to the reader.

As a clear, concise, well-arranged, text on the subject of direct current machinery this book deserves the highest praise.

J. H. M.

Electricity, Experimentally and Practically Applied. S. W. ASHE. New York, D. Van Nostrand Co. 12mo, 375 pp. Illustrated. $2.00.

In this small volume the author has endeavored to cover the field of direct currents and direct current machinery, the theory of alternating current circuit, and also the theory and operation of alternating current machinery. As must necessarily be the case in such an attempt, the subject matter is presented in a very elementary and superficial manner; another fault with this work is the rather illogical sequence in which different topics are introduced. Many experiments, illustrating the phenomena treated in the text are described in detail; a number of very good cuts materially assist the descriptive matter.

The book can never serve as a text-book for students of electrical engineering, but may be a very useful supplement to courses on applied electricity, such as one given today by various operating companies for their employees. It will also be appreciated by those who lack mathematical training but who are trying to acquire by themselves, some of the rudiments of the subject of applied electricity.

J. H. M.

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THE WEST COAST OF AFRICA.

BY FREDK. F. SHARPLESS.

The Gold Coast Colony lies on the southern coast of Africa or rather on that southern coast which lies to the north of the Gulf of Guinea. It has been known as a gold-producing country for several centuries, but until within comparatively recent years no attempt has been made by white men to do any systematic mining.

Active gold mining and exploration is, at the present time, limited to the English Colony of Ashanti and the Gold Coast Colony, an area lying between 5° and 11° north of the equator and with practically the same longitude as England.

Since active operations have been taken up by white men there have been a series of booms and bursting of bubbles, throwing the country at one time into the lime light and then into ill repute, the net result being that far more gold has been spent in the country during the past twenty years than has come out of it.

During the winter just past one of the periodic booms was on in London, and it was to learn the cause of this and to see just what the conditions were in the country that I went down and spent four months in looking over some of the operating mines and prospects.

In a general way I found that the banket deposits were receiving the greatest attention both in the field and on the stock market, while the quartz veins were scarcely receiving the attention they deserved either on the market or by private investors.

The investing public of London have learned to attach great weight to the word "banket." There is a prevailing idea that it is

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synonymous with the word "dividend" or at least that something good always goes with banket. This is because so many good things have come from the banket deposits in South Africa and the banket deposits of the West Coast are in many respects similar to the better known reefs to the south.

Naturally promoters find it easier to secure money for banket deposits than for quartz. As the promoter's business is to secure money he follows the line of least resistance, regardless of ultimate results, and promotes banket propositions.

Gold is found in three distinct classes of deposits in Ashanti; alluvial gold, in banket formations quite similar to those of South Africa as above noted and in quartz veins lying in an area outside of the banket deposits.

Alluvial deposits have been worked by natives over a large territory and by far the largest percentage of native production has come from this source.

Streams and benches have been worked extensively but I also found extensive areas where the gold had been derived from quartz veinlets in an easily decomposed rock, somewhat similar to the saprolite deposits of the Southern Appalachian system. The rock decomposing and washing away, has allowed the gold to concentrate to sufficient extent to make deposits workable by native methods.

A number of dredges are operating in the country but not with unadulterated success. The gravel is of fair grade but the deposits are generally small, requiring frequent moving of the dredges. The bars are full of buried timber. During the dry season the streams have but little water in them requiring to be dammed for moving the dredge. Labor for fuel cutting purposes is often scarce. The climate on the rivers is, of course, the worst in the country, so that altogether the dredge operators have much the worst of the economical conditions with which to contend.

The area of the banket deposits is unknown; I traversed it for a length of about 120 miles and for a width in several places of 10 to 15 miles.

The banket or bankets, for there are probably a number of parallel beds, consist mainly of quartz pebbles and sand usually cemented with silica. In some instances, however, the quartz pebbles were imbedded in sediment that now has the form of shale. The enclos-

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ing rock is sandstone or quartzite, which in turn rests on granites and is overlaid with slates.

At the southern end of the formation the quartzites occur in a number of folds apparently of slight elevation, exposing in cross section one or more beds of banket at several places. Whether there is really one or more beds of auriferous banket here has not been determined. Only one, the Tarkwa banket, has produced any mines, and I am rather inclined to the opinion that the other exposures are chiefly parallel beds lying higher up in the series.

At the most northern point examined, I am certain that there are two distinct beds, one of which I know contains commercial ore and the second probably contains ore of commercial value.

The beds at the most northern point examined are quite different in appearance from those at the southern end, and while this may be only a local difference, I think it probable that they are distinct from the deposits being worked at and near Tarkwa and are merely parallel beds of the same series of formation.

A peculiar distinction from the South African banket is that on the West Coast the bankets have little or no sulphides in them. What iron has been found even to the depth of 1,800 feet is in the form of ilmanite, which is supposed to be an original constituent or at least believed not to have been derived from pyrite.

Of hundreds of samples taken, I rarely found one of the banket to be absolutely barren, but on the other hand I sampled croppings in many places without finding commercial ore. Among the more important properties in the banket formations are the Effuenta, Tarquah, Abbontiakoon, Fanti, Abosso, and Cinnamon Bippo on the Tarkwa reef and the Wallis on the Wallis reef. These are not all producing, but they are either producing or in active stages of development.

The quartz veins lie to the northwest of the banket deposit. They are not all of one system but lie in several different geological formations.

The most successful mine of today, that of the Ashanti Corporation Limited, is working on quartz, milling from 6,000 to 9,000 tons per month, at a very handsome profit.

The treatment here is, drying, pulverizing, roasting and cyaniding. The roasting is essential because of the presence of graphite

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and because of much arsenopyrite which does not yield its gold until after roasted.

Another mine that will soon take a prominent place amongst the producers is the Prestea, Block A mine, also on a quartz vein.

The Prestea is a wide low grade vein (about $8.00 per ton), but by working 5,000 to 6,000 tons a month, a fair profit is expected.

The Bibbianni is another good but low grade property; being 75 miles from the railway it is not attempting to treat its lowest grade ores.

A great help to prospectors of today is the work done by natives within the past 300 years. For years the natives have been gatherers of gold, the greater part of which has been dug from shallow alluvial deposits, but there is also much evidence of quartz mining. Scattered over a wide territory are found the pits and small shafts of the black miners and by far the greater number of good ore shoots so far found by white men have been encountered below native workings. In fact without these guides development by Europeans would have been very limited in extent.

The area under consideration is all within a dense tropical jungle, prospecting is carried on under great difficulties. Croppings are rare and when found are usually of hard dense quartz of no commercial value.

In going over a new piece of ground every foot of the distance must be cut out before the jungle can be penetrated and then one can scarcely see 20 feet on either side of the path that has been cut.

If one attempts to take out life insurance before visiting the West Coast, he gets a suggestion of the climate that he is to encounter. The life insurance companies are not anxious for the risk at any price. Present conditions, however, do not warrant this attitude. The country is low and damp and malaria is ever present, but one may fully protect himself against this and most of the other troubles of the country. Five grains of quinine per day, boiled water at all times, a very limited use of alcoholic beverages and back and head well protected from the sun are the essential precautions.

Sun and an excessive amount of alcohol account for the majority of deaths in the country. Fevers are an incident but are seldom fatal. Frequent repetition of malarial attacks is liable to be fol-

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lowed by black water fever, and this coming on a constitution already weakened may quite possibly be fatal.

Yellow fever is probably always present along the seaboard, but there have been no bad outbreaks for a number of years and the mines will scarcely be molested by it except as it may close the ports of entry.

Government employees stay in the coast country twelve months at a time and then have four months off. A man engaged in active mining operations should not stay more than eight months at a time with four months off in each year. And if making a hard and active trip through the country without rest of consequence, four months is long enough.

With the dense jungle surrounding the mines one might expect the fuel question to be solved easily, but it is not. Much of the wood is unsuited for boiler or timber purposes, and it does not take long to clean out that wood which can be used to advantage, already several of the mines are importing Belgian coal at a cost of about $14 per ton at the mines. There are no water powers in the country and in spite of the climate being moist all of the time, water for milling purposes is often scarce.

Indications of oil are found along the West Coast and considerable drilling has been done but up to the present time no fuel oil of consequence has been discovered.

A government railway runs through the center of the mining country from Sekondi on the Coast to the town of Kumasi, about 160 miles to the north. Freights are very high and I formed the impression that the mines were worked for the benefit of the railway and government officials rather than that the railway was operated for the benefit of the mines.

Other than the railway all transportation is by man power, or woman power, and it forms one of the heavy items of expense of operating in that country.

Native labor is poor at the best and though the country is well populated only a small percentage of it cares for or is capable of the heavy work around the mines.

The negroes of the coast towns, the negroes who have been in contact with the whites for a number of years have acquired some of our extravagant habits and are sufficiently in need of money so that there is an incentive to work.

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The tribes from the interior, where nature has been less prodigal with its food supplies, furnish a fine type of men, they are strong, well built, intelligent and honest fellows, but so far these men have acted only as porters and have not been willing to go under ground.

The great majority of the natives can live off the land with very little exertion, and as slavery has been practically done away with, they will not work.

The native labor supply is not an unsolvable question, but at the time of my visit there was but one mine in the country with an ample supply.

Our countryman, Henry Hay, one of the best organizers of the Transvaal is now on the West Coast where he is trying to work out a practical solution of the labor and other economical questions for the Wernher Beit & Consolidated Gold Fields interests.

While I am far from advocating a return to slavery for this country it is evident that the sudden removal of restraint from all of the lower types has had a bad result in many respects.

Human beings of the lowest type who have been subjected to slavery for generations can not be expected to make good citizens capable of caring for their own interests in the short time of two or three generations, and if there were any great expectations in this direction they have not been fulfilled.

It will never be a white man's country and a hundred years of attempted christianizing has not had results commensurate with the efforts and money spent in that direction. There is no dearth in the number of converts that have been made, but most of the so-called Christians were morally and physically better men before the Christians came amongst them. It is simply the old story of acquiring all of the faults and none of the virtues of the white men.

The Mohammedans working down from the north have met with far greater success. The converts to this religion attempt to live up to its tenets, they understand it better, it is taught by men of their own color, men who are more in sympathy with the black man's mind.

The result is that when in search for honest and intelligent men, one hunts not amongst the natives but amongst the men from the northern countries.

A tremendous amount of the work of the country must however

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be given to the educated negroes native to it. Train operators, telegraph and postal clerks, clerks in the stores, time keepers and a score of those offices for which white labor would be too expensive are filled by native negroes with corresponding inefficiency as a consequence.

One of the trials of the white man is in his frequent intercourse with these inefficient and officious semi-educated negroes. Fortunately there are some exceptions but the English government and European employers will always have difficulties along these lines.

Let me say in conclusion, that the country has been maligned, while not an El Dorado gold ore of commercial value is of frequent occurrence, the climate is bad but not unendurable, labor is poor but it can be procured, mining and treatment costs will always be high but not prohibitive, therefore, when you are called upon to pass an opinion on a West Coast proposition do not turn it down simply because it is on the West Coast.

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GEOLOGY AND ECONOMICS.*

By JAMES F. KEMP.

During the century which has just closed, the various branches of natural science, botany, zoölogy, geology and their relatives, having earlier completed their childhood, attained to the well-rounded development of maturity. Their broad truths were given clear expression; they were widely apprehended; and they became the foundations of various inventions and applications of far-reaching influence upon human welfare. Geology, although closely bound up with agriculture, has nevertheless been especially concerned with mining. And justly so, because its contributions to the art of mining have been no more than a filial return, since mining as practiced in the Middle Ages was the parent of geology. Until recent years geology's services to the industry have been chiefly rendered in spreading sound and reasonable ideas regarding the nature and distribution of the useful minerals; in solving the perplexing structural questions affecting their occurrence; and in facilitating the discovery of new fields.

The problems of the production of the metals and non-metalliferous substances, as we know them today are of quite recent growth. High explosives, efficient engines and pumps, steam shovels and the like are all not so old as many men who are still living. They have so greatly reduced costs that practically a new world has opened to the miner. Not only on the surface or near it has he been able to work, but the depths have become accessible, and where the value of the ore justified the effort, no floods of water have sufficed to keep him out.

These successes coupled with ever expanding markets have until recently directed attention almost wholly toward discovery and production. But the last ten years have brought a further change. We are now less concerned about new discoveries than about the maintenance of old ones. We are not altogether intent on produc-

* Presidential address to the New York Academy of Sciences, December 19, 1910.

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tion but are much given to forecasting and husbanding. From being solely an aid to the miner, the active worker, the producer, geology has become the colleague and helper of the economist, the statistician, and the philosopher.

Like all other changes in fundamental points of view, this one has not come with absolute suddenness. As far back as 1879 certain geologists and engineers began to raise and discuss the question of the duration of the Pennsylvania anthracite. In 1894 the late Richard P. Rothwell, long the able editor of the *Engineering and Mining Journal* gave these coal fields a future of 70–100 years. Thus for over thirty years the question of their death has been a very live one. Even earlier the future of the coal-fields of Great Britain came up for discussion. A parliamentary commission was appointed in 1866 and reported upon the question in 1871. For forty years anxiety has prevailed regarding the continued production of our petroleum wells—and naturally so. The very means of production of this useful source of heat and light starts a train of thought along the lines of its permanence.

Some ten years ago, the question of our reserves in iron ore began to excite interest. Mr. Andrew Carnegie gave most forcible expression to the feeling of alarm in his rectorial address in 1902, at the University of St. Andrews, Scotland. Mr. Carnegie was known from one end of the world to the other as one of our greatest iron-masters and his words made a profound impression. In his address he assigned us only enough first-class ore to last for 60 or 70 years, and only enough of the inferior grades for 30 years thereafter. We all trembled for some years with the prospect of seeing our greatest industry in the production of metal disappearing within a century. Many thoughtful people began to wonder what would become of us with its extinction.

I have thought, therefore, that it might be not without interest if we take up this evening the more important of our metals and pass in review some of the fundamental facts of their production, the yield of their ores, the foreign sources, the future probabilities and the effect upon the civilization of our own and other lands which would result from their curtailment. In a word we may for a time discuss geology and economics.

The iron industry in the United States took its rise in the colonies

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along the Atlantic seaboard, and at the outset was based upon the magnetic ores and brown hematites there occurring. For one hundred and fifty years its growth was slow. In the decade of the forties and fifties of the past century it had spread to the Adirondacks and in the fifties began its development in the Lake Superior region. Not until after the close of the Civil War and the resumption of peaceful activities did this great industry manifest its possibilities. With improved facilities of navigation which placed Lake Superior in easy communication with the coal producing states of Pennsylvania and Ohio, the iron ore producing states of Michigan, Wisconsin, and later Minnesota came rapidly into prominence. In somewhat slower growth Alabama, during the seventies and eighties gathered headway. At present four fifths of our ore supply comes from the three Lake Superior states, and three out of the four-fifths from Minnesota alone. Alabama, Tennessee and Georgia together yield one tenth and the remaining tenth is divided among a dozen or more other states of which New York is the leader. Since 1880 the total has increased about seven fold and Pennsylvania, then the source of about one quarter the supply now yields approximately one and one-half percent. Minnesota, now the great source of ore, only entered the lists in 1884, and only began to utilize its present great mines about ten years later.

Thus in the brief course of thirty years there have been very great rearrangements not only in geographical sources of supply, but still more in actual amount of output. In normal, prosperous years the annual production is somewhat more than fifty million tons of ore.

But there have also been other changes not less striking. In early days and in remote situations only the richest ores could be mined. Magnetites for example in the lump from the Adirondacks afforded over 60 per cent. metallic iron. Specular hematites from the Lake Superior districts necessarily yielded 65. For some years no one regarded them with respect if they yielded less. Red hematites from Alabama afforded forty-five to fifty. The minor ores near the furnaces were often much lower—but they may be passed over for the moment in emphasizing the larger features. Magnetites in the Adirondacks are now concentrated before shipping

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and in instances two and one half to three tons are condensed to one of 65 per cent. tenure. The crude ore carries 33–35 per cent. During the early years of the present decade the general average of Lake Superior shipments fell off about one per cent. per year—so that now the soft ores, so called in contrast with the hard lump specular of earlier days, range somewhat above 50 per cent. Alabama ores once forty-five to fifty now are very uniform at 36 to 37. So far as the brown hematites are concerned, which in the form of lumps, crusts, pipes, etc., are distributed throughout ochres and clays, the percentage of available iron in the crude ore is lowest of all. We wash from eight to ten tons of crude in order to get one ton of concentrates of say 40–45 per cent. in iron and under favorable circumstances may treat much lower raw materials. Soft magnetites in Pennsylvania, which on the richer outcrops gave 45 to 50 per cent. are now dug in very large amounts with a yield of 43. If we take the total production of ore in the United States and the total production of pig iron, we find the yield in the large way to be about 50 per cent.

In order to gain some idea of the comparative merits of these figures when set along side the yield of the ores produced in other lands, a few cases may be cited. Germany in 1907 produced 27,700,000 tons of ore, exported nearly four millions and imported eight and one-half millions. Of the local production three quarters were obtained from Elsass, Lothringen, and Luxemburg, whose percentage in iron ranges between 30 and 40 and is on the whole not very different from Alabama's present percentages of 36–37. Germany's imports of course range much above these figures, else the ore could not stand the freight charges from mines in such remote countries as Sweden, Spain and Algiers.

Great Britain yielded, in 1907, approximately 15,000,000 tons of which about three quarters were the so-called impure carbonates yielding 30–35 per cent. iron. One ninth of the total was red hematite at 50–55. The general average would be somewhat less than that of Alabama. Importations of richer ores especially from Spain helped to raise the furnace yield.

France in 1908 yielded 10,087,000 tons of which 88 per cent. was mined in French Lorraine of the same type as the main German supplies. The ore ranged from 33–40 per cent.—again not far

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from the Clinton ores of Alabama. We are justified therefore in saying that the largest part of the output of the next three producing countries of the world is about the same as the lowest grade of lump ore which can be profitably mined under present conditions in the United States. When, therefore, we come to estimate comparative reserves we must realize that in the Lake Superior region —our greatest producer—we pay no attention today to ores, which are nevertheless much richer than those of Great Britain and continental Europe.

In the opening sentences I spoke of the anxiety which was felt a few years ago regarding the reserves upon which the industry would of necessity rely for its future. I mentioned Mr. Carnegie's remarks in 1902 at the University of St. Andrews. But he was not the only one who discussed this question and now in referring to one or two other forecasts, I think you will have in mind some of the fundamentals which establish a correct point of view.

In 1905 Professor Törnebohm, the eminent and greatly esteemed former Director of the Geological Survey of Sweden, assigned to us a reserve of only one billion and sixty millions of tons. Obviously at an annual production of over fifty millions this reserve would only last twenty years. The future thus looked still darker than when seen through Mr. Carnegie's spectacles. Much opposition arose at once, however, to Professor Törnebohm's data, because from them had been omitted the red hematites of Alabama, which can be very accurately estimated and which of themselves are thought by competent observers to have at least half a billion tons for the future. Additional modifications must also be introduced when we properly appreciate the downward tendency of workable percentages. The lower the percentage of iron which we require in the product of our mines, the greater the amount of ore which at once becomes available. This is peculiarly true of iron, because of its very wide, general distribution.

In 1907 in anticipation of the International Geological Congress of 1910, which was to be held in Stockholm, the Swedish Committee of Arrangements began the preparation of a series of estimates of iron reserves in all the countries of the globe. Geologists familiar with local conditions were requested to prepare the figures each for his own country. It fell to the speaker to start the collec-

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tion of American estimates and much aid was afforded by several of the largest companies owning reserves. Shortly thereafter, however, the interest in the Conservation of Natural Resources sprang up and Dr. C. W. Hayes of the United States Geological Survey was empowered to use all the resources of this great organization in assembling data on iron. In this way figures as reliable as can be expected are now available. We learn from them that we may consider three and one-half billion tons of fifty per cent. ore as assured in the Lake Superior region. Of this great total three billions, one hundred millions are in the Mesabi range of Minnesota. At thirty millions of tons per annum, the present output of Minnesota, we have a reserve for a century.

On the other hand if we drop to 40 per cent. or slightly below, still however remaining a few per cent. above the Alabama grade, the drill holes show above depths no greater than those already reached in some mines, two or three hundred billions of tons of siliceous hematites, giving amounts practically inexhaustible.

In the Alabama ore beds we feel assured of five to six hundred millions tons of the grades now utilized and there may well be twice that number. The conservative estimate would afford enough to last at the present output of that state longer than a century. In addition there is much reason for thinking that there may be two or three times as much.

Speaking for the country as a whole, we may say that there is an assured and demonstrated supply, at present rate of output and at present percentage of yield for about a century. There is, furthermore, a less accurately measured but still very probable addition, when we allow for lower grade but still practicable ores, which will be sufficient to last for fifteen hundred years to come.

If, however, production increases as indeed it may with a rapidly growing population, and if in this way heavier and heavier drafts are made upon even this great reserve, where shall we look for more. There may be some new discoveries within the United States, but at present it is impossible to speak definitely of them. We may ask if there are other supplies in neighboring lands. To this question we may answer yes. Along the north shore of Cuba, toward its eastern end and near the sea, three areas of what formerly appeared to be a barren ferruginous soil have been dis-

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covered and tested, so that we now know that there are two to three billions of tons of a very pure iron ore, which, when deprived of the large percentage of water which it contains—a cheap and simple process—will yield from 40–45 per cent. iron. This variety of ore already begins to enter our ports and the deposits will undoubtedly contribute in no unimportant way to the output of our furnaces.

The report of the International Geological Congress has shown further that in Newfoundland there are quite probably more than three billions of tons of red hematite, whose present yield averages 54 per cent. From Brazil, moreover, in the state of Minas Geraes, but pretty well back from the coast and not yet opened up by rail, as estimated by Dr. O. A. Derby, there are from five to six billion tons of 50–70 per cent. ore awaiting the drill and the steam shovel. Ore from Brazil faces a long sea voyage but the grade is rich and the iron masters of this and other countries are looking upon these deposits as well within the possibilities of the future. Ocean freights are kept at very reasonable rates in these later days and once on a steamship even so low-priced a commodity as iron ore, if of good percentages and cheaply mined, can be taken relatively great distances. This is demonstrated by the shipment this year from the mines of Kiruna, 112 miles within the Polar Circle in Lapland, of 300,000 tons of ore, 113 miles to the Norwegian coast by rail, and over 4,000 miles to Philadelphia by sea, with no great prospect of a return cargo. These shipments also demonstrate that we are not without the range to which European ores may be shipped when exceptionally rich. Some portion of the vast ore body of Kiruna, with its demonstrated 500 millions of tons of 60–69 per cent. ore, will also reach American furnaces.

But even were our actual ores of present grade to become exhausted, iron as a metal would not fail. The basic rocks with their low percentages still remain. The trap-rock of the Palisades contains 7–8 per cent. of metallic iron, a value that is far above the general yield of our copper ores in the red metal.

Iron therefore will never fail. It will probably not change in its general relations to modern conditions for a very long time to come, so far as its ores are concerned. We may have greater anxiety about the supplies of coking coals than about the iron ore, but

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there are always such possibilities of improvements or changes in processes that no one can justly give way to unqualified forebodings.

Copper is the metal generally considered next in importance to iron. It is a very old one in the history of the race. The bronze age, you will recall, preceded the iron age. Prehistoric man in Europe solved the mixed metallurgy of copper and tin before he learned the smelting of iron. Prehistoric man in America found native copper on the shores of Lake Superior and passed it in trade a thousand miles from its home. As a cherished possession it constituted his ornaments while he lived and it was buried with him after he had died.

Among the moderns, copper is most extensively employed in brass, but as a conductor of electricity it finds year by year increasing applications in the purest condition in which the metallurgist can supply it. If at home or in your office you look around your chair or desk you will be surprised to find how universally employed it is.

Greatly stimulated by the development of electricity in later years the production of copper has advanced by leaps and bounds. At present the United States are the heaviest producers, with Spain following next but only yielding one eighth as much. The United States furnish over half the total. In 1850 this country yielded 728 tons; in 1900 over 303,000 and in 1908, 471,000. Meantime in 1850 the price of copper was about 30 cents per pound. Its lowest point in recent years was nine cents in 1894. Its highest, 25 cents, was attained in 1907. We may each of us imagine the variation in the profits of a mining enterprise as between 11 cents a pound and 15 cents, let alone 20 or 25 cents. Mining costs, smelting and freight charges show no such variation, so that with rising prices profits greatly increase. Indeed few of the metals have such extraordinary ups and downs as does copper.

In its ores the yield varies greatly. On Lake Superior where the native metal is distributed through ancient lava flows in little pellets, leaves and sheets, it has been profitably mined and produced through periods of years when it constituted but three quarters of one per cent. of the ore. The general run is, however, one per cent. and above. If we recall that in a ton of 2,000 lbs. one per cent. is 20 pounds, and three quarters of one per cent. 15 pounds,

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and if copper is selling at say 13 cents, the mining manager must break down, hoist, concentrate with attendant losses, and smelt an ore worth less than two dollars for all the metallic contents which it contains. We can thus gain an idea of the close and economical work required and the ability demanded of a manager. As the price rises the profits greatly increase, and temporarily idle mines are brought within the widening remunerative zone, so that they are quickened into life. As the price falls, the mines dangerously near the line, close down and production ceases. The lowest cost of production claimed is from the low grade and very large ore bodies of the west and is placed at or about eight to nine cents per pound laid down in New York.

In copper ores outside of the Lake Superior region, we usually find the metal in composition with sulphur. The ores as they come from the mine may be rich enough to go directly to the smelter, or they may require concentration before the grade is sufficiently high. The ores which are directly smelted reach the minimum of copper in the Boundary district of British Columbia, but associated gold and silver raise the value per ton above four dollars. Copper ores yielding copper alone were smelted at Ducktown, Tenn., during long campaigns at a little less than 2.5 per cent. In earlier years and in many mining districts ores as high as 20 per cent. were found, rarely even higher, but they in time were exhausted and five per cent. would be quite rich for day in and day out averages.

These statements will serve to establish a point of view and likewise afford a standard of comparison. What is the outlook for the future of copper production?

We cannot predict copper with the certainty of iron. It seldom appears in bedded deposits which can be measured. In the deep mines we cannot always see ahead for more than a year or two. In some mines we know from exceptionally complete development, of twenty years supply. But the great advance in copper mining has been the entrance of relatively low grade ores into the productive field. The wall rocks of ten years ago have become the ores of today. Where we find in porphyries or schists copper sulphide disseminated in fine particles, or as coatings along crevices and in sufficient richness to yield two to two and one half per cent. throughout very large bodies, it can be mined very cheaply and

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concentrated in enormous quantities so as to return a safe margin. If the ore lies near the surface, steam shovels make excavation extremely low in cost. The huge pits and open cuts of this type of mine in the West are now among the great sights for the traveler. Mills whose insatiable crushers take as much as eight or ten thousand tons per day are no longer unknown. The drill blocks out the ore long before mining begins, and reserves can be estimated more closely than in the vein mines.

If a mine is called upon to furnish a mill with 2,000 tons per day and we allow 300 working days in the year, 600,000 tons must be supplied per annum. For a life of twenty years, a time practically demanded of such an enterprise to justify the great expense of installation, at least 12,000,000 tons must be shown by the drill before the enterprise can safely begin. If we expect to mine three times this amount per day we call for three times as much ore. These figures large as they may seem are not beyond the estimates of ore bodies as now blocked out in several places in the west, and even with these great demands, twenty years supply and even more in instances have been demonstrated.

Let us now imagine again a 2,000 ton daily output of say 2¼ per cent. ore, of which the mill saves two thirds, or 30 pounds of copper in the ton. The output in copper per day will be 60,000 pounds, or 30 tons and for the year 9,000 tons. Should three new companies start up with four or five times this output, 36,000 to 45,000 tons will be added to a yearly supply which in 1909 was 552,668 tons. We see great need of a growing demand in order that these vast contributions may be absorbed. Yet I have made no unreasonable assumptions nor have I overstepped the practical certainties of the next few years.

How long will our copper hold out? Mines come and go and for the immediate future there will certainly be no scarcity. Copper does not oxidize as readily as iron and is not lost. The world's stock steadily accumulates.

But twenty years is not a long look ahead. Are there new countries which will be producers? Some of the old mines in Europe are now no longer great sources of the metal.

We do know of possibilities in Alaska that will add some contributions. We know of new or recently opened ore bodies in Peru,

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Bolivia and Chile that promise well. We hear of very large deposits in the southeastern corner of the Congo State, once worked by the ancients, now revived by the moderns and possessing large reserves of 15 per cent. copper ore. The Cape to Cairo railway will give them great impetus. For the immediate future there is no lack, but if we look fifty years or a century ahead we can speak with less confidence. In a general way we may say that probably new discoveries will for a time at least more than keep pace with demands; but when we look fifty years into the future we are not so certain. It behooves the producers to use no treatment of an ore except a careful and economical one. If tailings and waste from our mills now contain one third the copper in the original ore, they should be impounded and kept from being washed away by floods, against the possible call of the future. We dare not say that they will never be within the ranges of profitable treatment even though their low percentage places the copper beyond reach to-day. The copper situation is not one to excite anxiety, yet it is also one not to encourage extravagance.

Following copper we may take up lead and zinc, which are the next metals in amount of production. Of the three, zinc is the least in total tons and in total value. We may gain some idea of the relations from the small table given below in which zinc is taken as unity and the figures relate to 1908.

	Amount.	Value.	Price per lb.
Zinc	1	1	1.0
Lead	1.6	1.45	0.9
Copper	2.46	7.0	2.8

Thus we see that the lead production is one and three fifths that of zinc, and the copper is two and one half times; that the lead is about one and one half times the value of the zinc, and the copper is seven times; and that zinc is worth more per pound than lead and only about one third as much as copper. The red metal is not only produced in greater amount, but is worth more per pound and in the aggregate than both the others taken together.

Among the nations of the world the United States has become the chief contributor of lead and yields year by year proportions varying from 27 to 33 per cent. of the total. The next country is Spain with about two thirds as much, and Germany follows with three fifths.

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In this country the State of Missouri is the heaviest contributor and is responsible for practically 40 per cent. of the total. Idaho is next with about 32 per cent. and Utah follows with 13 to 14. The western lead all carries silver. The precious metal is an important factor in the value of the product. When we come to forecast the future it is not possible to see more than a few years in advance—or to speak in more than a general way. The miners would be glad to be assured of reserves of ore for a goodly period of years, but it is seldom possible or practicable to demonstrate their presence. Operations necessarily continue with a few years supply blocked out in advance of the actual mining and the hope is maintained that more will be found. Very often the expectations prove justified. We may therefore in a measure forecast future experience somewhat by the past. In the Missouri lead region mines have been operated for forty or fifty years, not on so large a scale at the outset as now, but continuously. For some years at least no change may be anticipated. In Idaho the lead ores are now known to continue to depths of nearly 2,000 ft. beneath the surface overhead, and to be holding out without essential change in character. In Missouri, however, the mines never have been very deep, that is, over three or four hundred feet and the compensation comes in wide horizontal extent.

Some of the old time heavy producers have greatly declined. Nevada once an extremely important source of lead is now a comparatively small contributor. Colorado, in former years our chief source, has dropped to only a third of its one-time yield, and yet the total of the country has gone quite steadily on. The fall in the price of silver was a hard blow to the western lead miners and naturally cut off not only their profits but raised the necessary percentage of metal in the ore.

If we look ahead for a century or some such long period, we may not feel assured that production can be maintained at present rates. There may of course be new discoveries in lands not as yet fully explored. Being distant from present centers of consumption as they necessarily would be, their entry into the markets would imply higher prices so as to meet the charges of freight.

On the other hand lead is a metal which oxidizes or changes very slowly. In its uses in the metallic state it tends thus to accumulate

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unless lost in use, as in the case of shot and bullets. It is extensively employed in the manufacture of paint and in this form is of course never recovered. About two per cent. of the entire output is destroyed to give us white and red pigments.

It behooves us on the whole to be careful in the use of lead and to avoid when possible its unnecessary sacrifice.

Zinc is a metal of comparatively late introduction into commerce in the large way. Although known for centuries it has found its chief applications in the last sixty years. There was no zinc mine in the United States until approximately the year 1850, and from the Missouri region whence we now obtain our chief supplies, the really serious contributions began about 1870. Lead, indeed, was mined and prized long before this, but the associated zinc ore was thrown one side on the dumps. In the West the same experience continued until much later. Zinc was a nuisance in the metallurgical treatment of lead and even the lead was sought and smelted either because of its own silver contents or because it made possible the treatment of other refractory silver ores. In the metallurgical work the zinc was volatilized or slagged off and was lost. Indeed one of our most serious metallurgical problems has been the successful treatment of lead-zinc ores and many investigators have addressed themselves to its solution. Now that anxiety is beginning to manifest itself regarding zinc supplies for the future, the desire to save it is stronger than ever.

Zinc, however, is a peculiar metal and because of the exigencies of its treatment its ores must possess greater richness and greater purity than those of other base metals. Thus in the case of copper a ten per cent. ore is in later days phenomenally rich and as it can be smelted in a shaft furnace the presence of iron or lime or other bases that make fusible slags is an advantage. But zinc ores, perhaps after preliminary roasting must be reduced and the metal must be volatilized at a high temperature from a small charge in a retort. The presence of fusible bases destroys the retort and the bases are therefore debarred beyond certain small percentages. Thus it happens that a forty or fifty per cent. zinc ore might be valueless if contaminated by iron or lime beyond a narrow margin. While almost any conceivable mineralogical aggregate that contained ten

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per cent. of copper would be a very valuable ore, a zinc bearing aggregate with four or five times as much zinc might be unsalable.

Suppose we compare them from another standpoint. Copper ores if at all profitable are worth about so much per unit of copper, that is, so much for each per cent. While there is some variation yet the contrasts as among three per cent., five per cent. and ten per cent. ores are much the same as the ratio of the per cents to each other. But if we think of a zinc blende ore or concentrate of 60 per cent. as the standard of richness, a fifty per cent. ore is not worth five sixths as much, nor a forty per cent. ore two thirds. On the contrary a forty per cent. might be entirely unsalable. As the zinc decreases other deleterious bases take its place and a worthless mixture soon results. Zinc is in many ways the most peculiar of the metals and when we come to deal with its profitable treatment, analogies with other metals fail.

In 1907 the United States were the chief producer of zinc among the nations but as a rule Germany leads, followed by this country and Belgium in the order named. In later years our output has varied from 26 to 30 per cent. of the total. As a rule Germany is 2–4 per cent. in excess of us and Belgium is 4–5 per cent. less.

In America, Missouri is the chief source of zinc. Its production from the mines was in 1908 approximately one half the output of the entire United States. New Jersey follows with somewhat over one quarter the total, while all the rest are much smaller.

The Missouri ores as thus far produced have been obtained from comparatively shallow depths. They extend lengthwise and sometimes laterally to greater dimensions than vertically. While it is not beyond the possibilities that lower lying deposits may be discovered, since zinc ores are found in Arkansas in strata of lower geological position, anticipations of this reserve have not as yet been demonstrated on a large scale. Kansas, Oklahoma and Arkansas, the states neighboring to southwest Missouri also have some zinc ores, but they are not of great importance; southwestern Wisconsin is a very old mining district and has many small mines, which were earlier worked for lead. They have been revived for zinc in later years and are now an appreciable but not great factor. They may develop somewhat more extensively and may last for a

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goodly series of years, but the mines are relatively small and are wet, so that exploration does not go very far in advance of mining.

In New Jersey the future is best forecast of all. For thirty or forty years there is no occasion of anxiety. Yet thirty or forty years pass quickly and then we must prepare to look for other sources. To make the zinc blende of the Rocky Mountain region available, an increase in price is practically necessary, otherwise the metal cannot stand the freight charges. There is zinc ore in the West but to what extent we cannot well say. It has been avoided rather than sought in most of our mines. Yet we do note symptoms of attention to it. In Butte, Montana, efforts are being made to concentrate it. Shipments of oxidized ores have been made from New Mexico for some years past. Large amounts, of peculiar appearance, seem to have been hitherto overlooked at Leadville, Colorado. They now promise to be an important resource. A government commission has reported on the occurrence of the metal in British Columbia in the hopes of utilizing the ores. From Mexico too we learn of explorations for zinc. Conditions are changing in the case of this metal and more and more it is certain to be brought from remoter localities. But when we look a long way ahead, say for a century, we cannot feel free from anxiety. This condition of mind is even more prominent in Europe than in America. The waning of the famous old mines near Aix la Chappelle, and the apprehensions felt regarding other sources, have led to a world-wide search. Zinc ores for example now reach Hamburg from the Pacific shore of Siberia, and as other discoveries are made, additional points remote from present smelting centers are likely to be shippers, provided that transportation is by water. Nevertheless, all these new conditions call for advances in price and before many years zinc bids fair to take the upward course.

The precious metals, silver and gold, are the only other two which we may pass in quick review. Silver is a less attractive object of mining than it was twenty years ago, and yet with the improvement of processes of extraction and with the great development of the output of copper and lead in which it is a by-product, the fall in its price of the early nineties has been less disastrous to the amount produced than one might suppose. Our maximum output was reached in 1892 when it was 63,500,000 ounces valued

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at $55,662,500. In the same year about 1,600,000 ounces of gold were produced valued at somewhat over thirty-three millions of dollars. In 1908 we are credited with approximately fifty-two and a half million ounces valued at twenty-eight million dollars. Gold meantime with the fall in silver, has advanced to 4,574,340 ounces, valued at $94,560,000.

In the United States we have now comparatively few distinctively silver mines. Among them Tonapah, Nev., has been chief. Mexico is the particular home of silver but the remarkable district of Cobalt, Ontario, has given great present importance to Canada. In our own country we must expect the white metal to share the fortunes of the copper and lead with which it is chiefly produced. As influencing its future, copper is a more serious factor than lead, both for the reason that Missouri lead contains little if any silver, and because western copper ores display greater reserves than do western lead ores. As sources of silver there were in 1908 no very great differences among Montana (a copper-silver state), Colorado (a silver and lead-silver state), and Nevada (a silver state). Utah (both a lead-silver and a copper-silver state) afforded about five sixths Montana's output; and Idaho (a lead-silver state) about three fourths Montana's;* Arizona (a copper-

Montana	10,356,200
Colorado	10,150,200
Nevada	9,508,500
Utah	8,451,300
Idaho	7,558,300
Arizona	2,900,000

silver state) follows after a long interval, and the others are much smaller.

As an indication of relative magnitudes, while the output of the United States was placed at 52.5 million ounces in 1908, Mexico afforded 72.6 and Canada 22 millions. Australia with 17.3 follows and then Peru with 7.2 millions. A metal with so high a value as silver will stand transportation from remote points and although the production in one country or another may fluctuate, the world's supplies are not likely to be seriously affected for many years. Silver is largely used in the metallic state, and being resistant to

* In ounces they range:

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change it tends to accumulate. Photography is the most destructive industry to it, and when once employed in this art, it is practically lost.

Gold is mined for itself alone to a far greater degree than is silver. Thus in this country in 1908, almost 93 per cent. of the gold was produced without regard to other metals and only 7 per cent. was obtained with copper and lead; whereas about 60 per cent. of the silver was produced in association with the base metals. Gold in later years has increased in amount of production beyond all previous experience. The steady and scientific digging and washing of low-grade gravels is in the long run more productive than the rich skimmings of the early California, Australia and Klondike placers. The world's total of 444 millions of dollars worth in 1908 was in excess of any previous year. The Transvaal furnished the most, nearly 146 millions. The United States followed with 96 millions; Australasia yielded 72.5; Russia nearly 40; Mexico, 24.5; Rhodesia, 12.2 and British India, 10.4. All the rest were under 10. The countries mentioned supply about 90 per cent. of the total.

In the United States 28 per cent. of the gold comes from gravels and these are the least permanent of the sources of the metal. With their exhaustion the output will decline. In the deep mines there are signs of waning output in some districts. In our own country new districts have come to the front from time to time to give on the whole a steady increase in output for forty years past. So far as the future is concerned, however, the ups and downs of any one or of several countries make slight difference in the world at large. Gold can be readily shipped from point to point and the place of its production is a comparatively small matter.

Like silver and to an even greater degree it resists chemical change so that the world's stock constantly augments. No very important portion is permanently lost in the arts.

Gold and silver are so extensively employed in coinage that they have received more attention at the hands of economists than any other metals. Gold in later years, with its increasing production has led to much philosophical speculation. The establishment of it as the monetary standard and the elimination of silver from this position have occasioned some of the most heated political

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controversies in the history of our country. Into these a geologist is not competent to enter. We all probably realize from old time experience how easy it is to become befogged. But the geologist can say that for some years to come the gold production will undoubtedly be maintained. And that while the Klondike and Alaska may wane, Siberia will increase.

We may now briefly summarize the main facts affecting the six metals which have been passed in review. It will then be possible to draw some general conclusions. Of iron ore there is no lack, nor need anyone be apprehensive regarding the supply of this metal, but before very many years have passed the yield of the ore will have decidedly declined. While the falling off will be gradual it will undoubtedly tend in the long run toward forty per cent. This change is in itself important because, unless otherwise neutralized, it will raise the cost of production. It makes necessary the melting of more barren materials in the furnace, so that the consumption of fuel rises with respect to the amount of iron produced. It means also the mining and freighting of an additional burden which yields no return. From whatever point of view we regard it, other things being equal the cost of production rises. The great reserves of lower grade ore than at present mined are in the Lake Superior district. They are siliceous ores, and will require in smelting the admixture either of limestone or other iron ores high in the bases. The Clinton ores of Alabama are of this type and except for the hitherto unfortunate percentages of phosphorus which they might add to Lake Superior ores, they would doubtless make an advantageous mixture with the latter. But the southern ores are remote from the northern. In order to meet them at or near the supplies of fuel a long railway haul would be necessary. While this is not impossible it would add to the cost so greatly as to be highly improbable. There is one further consideration. The greater part of our pig iron is used in the manufacture of steel. For this purpose in the two processes until recently most extensively employed, we need respectively either a very low or a fairly high percentage of phosphorus. If our irons are in between, and like the church at Laodicea, neither hot nor cold, they are ill-adapted to steel manufacture. Unless the growth of the open-hearth process introduces great changes, the

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mixture therefore of southern basic ores and northern siliceous ones is not altogether promising for this reason.

The greatest cause of apprehension as regards present processes of iron manufacture lies in the supply of coking coal. We have built lofty furnaces, and in their use we place upon the fuel as it progresses downward in the furnace a heavy load of overlying ore and limestone. We need a very strong coke to stand up under the burden. The coals which yield these high grade cokes are found in a small portion of the total coal-bearing area, and the life of the supply is one of the very serious phases of the present situation. I do not know what the amount of reserves may be.

While these physical and chemical factors operate to increase costs, there is always the possibility of improved processes and of greater efficiency to keep them down. The improvement oftenest in people's minds today is the utilization of water powers to generate electricity which in turn may supply heat. Now, in a blast furnace smelting iron ores, one third the fuel is employed in reducing the iron oxide and two thirds in developing the necessary heat for the reaction. Were we able with water powers to economically furnish electricity and from it derive the necessary heat we might save the two thirds of the present amount of required fuel. We might reduce costs. The remaining one third of the fuel we should always need but it is possible that poorer grades than high quality coke might answer. The saving would lie of course in the difference between the cost of the fuel and the cost of the electric current, provided the latter could be furnished more cheaply than the former.

The water powers in our own country or at least in the more thickly settled portions of it, have not failed to attract attention, nor have they gone altogether unutilized. The more conveniently situated ones are already harnessed to the dynamos. But in countries like Norway and Sweden, where there are large water powers still available; where there are rich deposits of ore; and where coal fails, the applications of electricity to iron smelting are likely to be first worked out successfully. Data may be furnished in the lifetime of many of us, which will cast light upon these improvements in their world-wide relations.

The only other apparent possibility of reducing costs lies in the

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labor charges. Wages at present are not unduly high, and unless the increasing population of the country brings to pass an inevitable struggle for existence, which will cause the greater subdivision of tasks at lower proportionate returns; or unless the general reduction of expenses for subsistence makes lower wages possible, there would seem to be slight prospect of change in this item. In any event the reductions from this cause cannot compensate the falling off in the yield of iron as foretold above.

Suppose iron goes up in cost—other conditions of our daily life remaining the same—transportation and all manufacturing based on machinery would become more expensive; and less freely carried on. Undoubtedly an appreciable pressure would be developed to turn our people back to the rural districts and to tilling the soil for a livelihood. The tendency under the stimulus of manufacturing development has been the other way. The migration of late years has been toward, not from the cities. Shall we perhaps find in the long run, in the increasing cost of iron and steel a partial solution of a much vexed problem? Will the cry, "back to the soil," receive support in a way not generally anticipated? The question is an interesting one for speculation.

The general inferences regarding copper are that the pinch of higher cost of production will be felt sooner than in the case of iron. We have no knowledge of such enduring reserves of copper ores as we have of iron. On the other hand, copper, despite its vast importance is not the fundamental necessity that is iron. It is used in less quantity in machinery and its increase in cost would less vitally affect manufacturing industries based on machinery. Advancing cost would cut it out of much ornamental work of inferior esthetic merit. The most serious effect would be found in raising the expenses of service in the applications of electricity. Electrical transportation, telegraphy and telephony would be more expensive than today. Unless wireless methods of transmission eliminate copper, or unless some discovery in the domain of physics which we do not now foresee, furnishes a substitute for the omnipresent copper wire of today, we may find ourselves face to face with some curtailment in these modern aids to the easy conduct of life's affairs. If in the course of several centuries the falling off in supply and the growth in population should raise

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copper to relatively high figures, we may wonder if a return in a way to the conditions of the Middle Ages will not result. Will copper then become to a greater degree than now the basis of skilled handiwork? Will the by-gone craftsmanship be revived and with a lessening total output shall we see an advance in artistic skill? In fact, if the vast development of machinery and the huge output of metallic objects at low cost—a condition so characteristic of today—should be checked or curtailed, would not hand-work on more valuable mediums of expression be restored. It is not altogether unreasonable to anticipate fewer objects and higher craft in their production.

The cases of lead and zinc are even more emphatic than that of copper. We have still fewer assured reserves and the pinch of increasing cost may manifest itself at an earlier date. The two metals are not, however, quite such vital factors in modern life as is copper and the larger effects would be less apparent. Zinc is a necessary component in the manufacture of brass the industry which absorbs the greater part of the copper output. A curtailment of either lead or zinc would cause inconvenience but would scarcely occasion fundamental changes.

Silver will be very seriously affected by a decrease in the output of either copper or lead. Gold will feel these changes in an appreciable but far less degree. There will always be sufficient, however, of each of the precious metals for coinage, and beyond this use their applications, except perhaps in photography, concern luxuries rather than fundamental necessities. We cannot attribute to them any profound possibilities in their influence upon civilization should the contibutions of the mines decline. In the recent past we have been more apprehensive regarding a too great supply of the precious metals, than regarding one too small.

With the increasing interest in the discussions of the conservation of natural resources, there has been an increasing disposition of the authorities to assume supervisory powers over mining and metallurgical operations. The old time idea that to the crown, or as we are accustomed to say, to the state belongs the mineral wealth of the earth, is experiencing something of a revival. The disposition to restrict the waste of valuable metals or minerals in processes of manufacture is commendable and after careful demonstra-

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tion that it is feasible and that it is just to the operating concerns, it may be wisely done. The rejected product of concentrating mills or the "tailings" so called, when provided with appreciable percentages of metals, may well be stored where they can be utilized by future generations, if processes improve so as to make them available. That is to say, they should not be run into rivers, or placed where they will be dissipated. The same remark applies to slags from metallurgical works. The moderns, for example, are now working over the lead-bearing slags left by the ancients at the great lead mines of Laurium, Greece. Even the slags of early smelters in the West and Mexico may again pass through the furnace.

Another question relates to the discovery, location, and ownership of mining property. So far as the metals are involved, and with the metals this address has been alone concerned, the valuable discoveries are so few in comparison with the disappointing attempts to develop, that only by encouragement and rather generous conditions will the prospector be enabled to follow his arduous calling. He must be offered large prizes proportionate to the many failures. He must be assured of possession by a very circumspect and conscientious administration, if confidence in the justice of the government is to be maintained. People in the parts of the country were mining for the metals is not carried on, hear only of the great successes and little of the innumerable disappointments. Far the largest part of the population thus acquire very distorted views of the real conditions of mining. The interference by the government other than in the ways which I have mentioned and in maintaining reasonably safe conditions for the workman, is a matter to be regarded with great caution, lest irreparable injury be done to the large problem of maintaining our future supplies with such new discoveries and developments as may be feasible. The wisest course is to improve the method of establishing and recording titles to new discoveries, and then, except in the matters already mentioned to let the natural course of business assert itself. The proper share of the state will be obtained through the normal processes of taxation.

The mines for the metals do not however present the most important phase of this subject. Coal is a more serious problem,

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and one demanding more extended treatment than would be justifiable in an address primarily devoted to other themes. One may only express the hope that where cases of dispute arise they may be determined by the courts, according to the established rules of evidence.

The resources in the metals which have been found in the United States have proved so great as to make the industries based upon them a very vital factor in our whole civilization. Great changes in the supply or the cost will inevitably react in the long run upon the opportunities for employment and support, and upon the very nature of our national life. While it will be a long time before re-arrangements in the case of the most important of the metals, iron, will be manifest and while they will assert themselves gradually, we are quite certain to face new conditions in copper, lead and zinc at an earlier date. In the end, however, we can perhaps justifiably forecast a future in which agriculture will figure more and more prominently and in which the moral, intellectual and spiritual life of the nation will readjust itself accordingly. Great and concentrated wealth is likely to be less in evidence, materialistic influences less pronounced, and from the vantage ground afforded by the greater comforts and opportunities of modern life as compared with that of a century or a half century past, we may in the distant future look forward to an evolution upon somewhat different lines. Broadly viewed the national life will probably be increasingly sympathetic with art and with ideals.

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DISPOSAL OF CITY WASTES.

By PELL W. FOSTER.

The problem of sanitary and economic disposal of the solid refuse of cities has recently been urgently presented to the cities of this country.

Several large cities are testing out the English system of destruction by fire in specially designed furnaces, and the purpose of this article is to describe in a general way one of these furnaces and the results of tests run with the refuse of an American city.

Refuse consists of three parts:

The ashes, which contain about twenty-five per cent. of combustible and seventy-five per cent. of non-combustible matter.

The rubbish, consisting of paper, rags, wood, iron, etc., which has considerable calorific value.

The garbarge, or kitchen wastes, such as fats, vegetable matter, etc. This contains about seventy per cent. of water, so that it has a negative calorific value.

The problem is to burn as much as possible of the combustible in this heterogeneous mixture without additional fuel, and to maintain thereby such a constant temperature as will entirely destroy everything organic in it, so that the resulting gases shall contain nothing offensive and the resulting solids shall be absolutely innocuous.

I describe below a 300-ton Heenan Destructor installed last summer for the city of Milwaukee.

As will be seen from the photographs and drawings the plant consists of four separate independent units, each of 75 tons daily capacity, arranged symmetrically in the four corners of a rectangular two-story and basement building. The principal work, such as clinkering and operating the plant, is done on the ground floor level, where the superintendent's office, electrical machinery and recording instruments are located. The upper story is used exclusively as a feeding floor and for storage of refuse, which is hoisted into the building and distributed into the various storage

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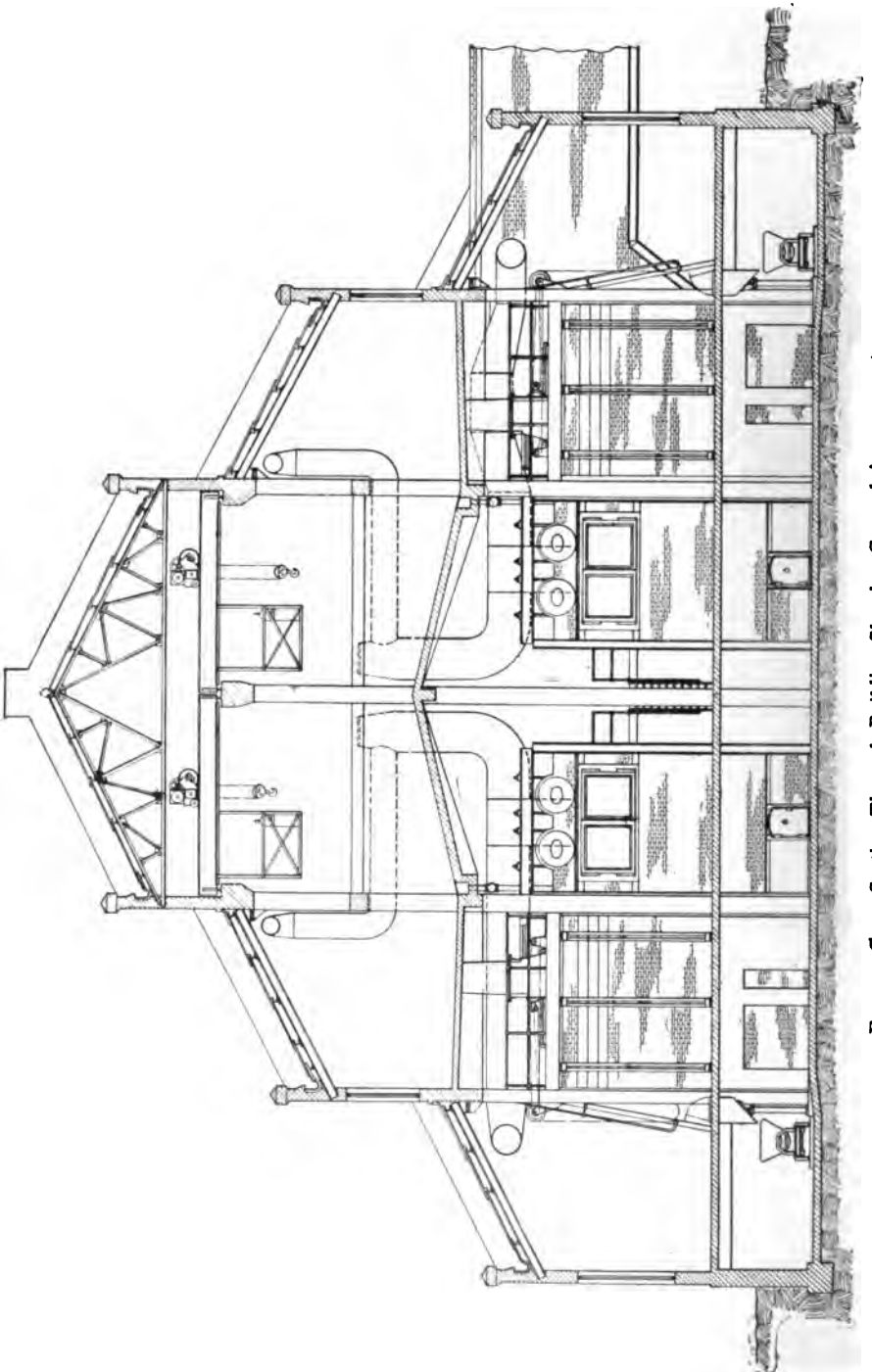

FIG. 1. Cross Section Through Building Showing General Arrangement.

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bins by a pair of over-head electric traveling cranes. The basement is used for clinker railway, doors for cleaning the dust from the furnaces, and the men's wash-room.

The refuse is collected by the city and received at the plant in two classes of carts. The one-horse carts have removable steel bodies, about 1½ cubic yards capacity each, and are used chiefly for collecting garbage. These car bodies are hoisted by the cranes

FIG. 2. Clinkering Floor.

and their contents discharged into bins on the feeding floor. The two-horse wagons are bottom-dumping of about 2½ cubic yards capacity. They are used chiefly for the collection of rubbish and ashes and dump into boxes resting in pits at the ground level; the boxes are picked up by the cranes and their contents thus transferred to the storage bins. From the storage bins, the men who feed the furnaces rake or shovel the material into the mouths of rectangular steel chutes termed "containers." These containers are located directly over the top feed openings into the furnaces, the

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sliding doors to the furnace openings also serving to keep the bottoms of the containers closed. The man feeding from above keeps the containers charged with mixed material at all times and the doors admitting the contents into the furnaces are controlled by the fireman from the floor below. The material when dropped into the furnace by the operation of the sliding doors, falls on a drying hearth where a large proportion of the moisture is carried away with the hot gases. The fireman rakes the dried material forward on the grates as required, for which purpose small stoking doors are provided. After a sufficient amount of material has been

FIG 3. Feeding Floor.

burned on each grate to form a thick body of clinker, the large counter-weighted clinkering doors in the front are opened and the clinker after being broken up by bars is drawn out of the furnace and dropped through holes in the floor into steel dumping cars on tracks below, whence it is pushed outside the building and allowed to cool on the ground.

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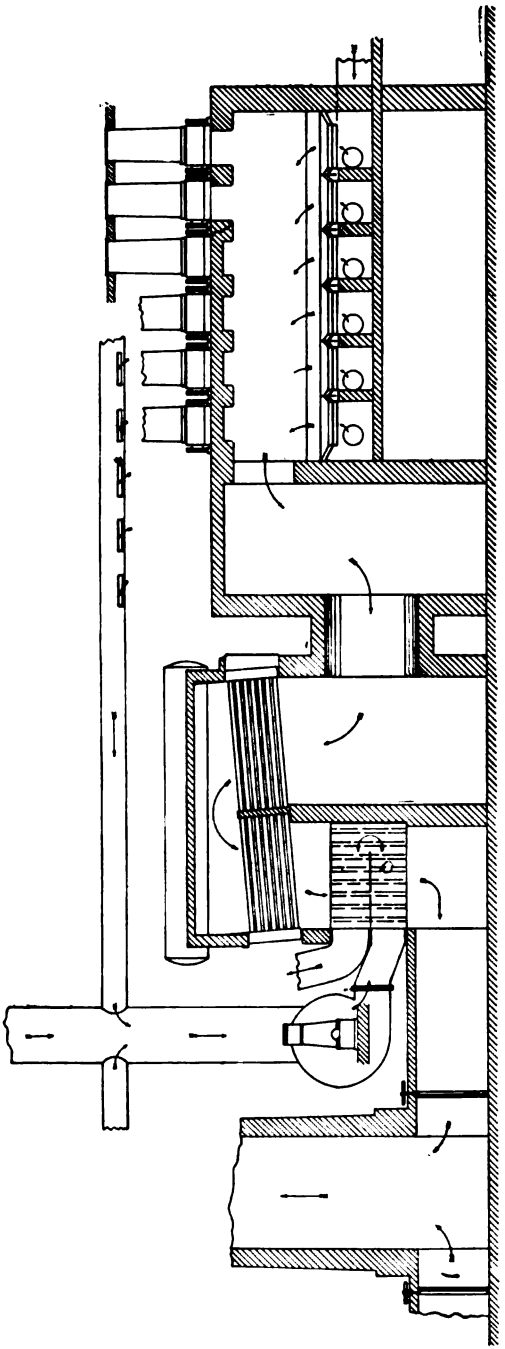

FIG. 4. Diagrammatic Sketch Showing Path of Gases and Air Supply from Ventilating Ducts.

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Each furnace consists of six separate grates located three on each side of a central combustion chamber through which the products of combustion of all six grates pass and are thoroughly mixed. The velocity through this combustion chamber is so slow that the combustion is entirely completed and dust is allowed to settle. After leaving the combustion chamber, the gases from each unit pass over the heating surface of a 200 H.P. water tube boiler, and then through an air heater or regenerator on their way to the chimney. The combustion is entirely controlled by forced draft,

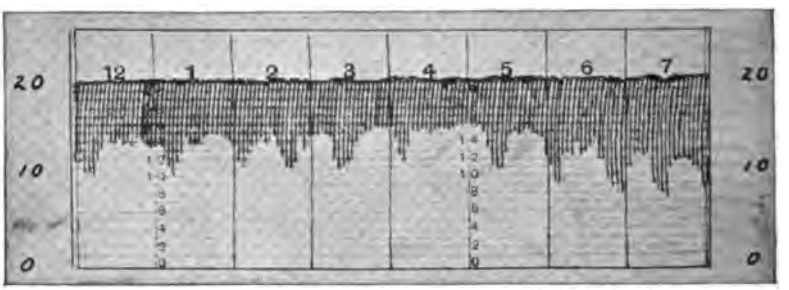

FIG. 5. Average Annual Refuse. May 27, 1910. 12 m. to 7.45 p. m. Average percent. $CO_2 = 13.1$. Sample 6. CO_2 Records, Official Tests.

all the air being drawn from ventilating ducts in the building by an engine-driven fan and raised to a temperature of over 300° in the air heater. The proportions are such that the amount of air required for combustion when the plant is operating at full capacity is seven times the cubic contents of the building, which insures the renewal of all the air in the building about every eight minutes, and prevents the escape of any gases from the building except through the chimney. Separate ash pits are provided under each grate with controlling valves, so that the rate of combustion and the amount of air supplied may be varied under the individual grates at will. Each unit is fitted with its own fan and engine with independent flues.

According to the terms of the contract the plant before being turned over to the city was officially tested with mixtures of refuse representing average annual conditions, extreme summer conditions and extreme winter conditions.

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AVERAGE ANNUAL MIXTURE TEST.

Composition: Garbage 41%
Ashes 41%
Rubbish 5%
Manure 13%
Date of Tests: May 26, 27, 30, 31, June 1, 1910.
Quantity of refuse consumed in 37 hours..... 126.81 tons.
Rate of burning per sq. ft. area..... 64 lb.
Steam evaporated from and at 212° F. per pound refuse destroyed ..... 1.34 lb.
Temperature of combustion chamber: Average..... 1,664° F.
Minimum..... 1,267° F.
Temperature of air supply ..... 351° F.
Cost of operation:
Labor per unit of 75 tons rated capacity per 24 hrs.:
One feeder, 37 hrs. at 25c. .....$ 9.25
Three firemen, 37 hrs. at 25c. 27.75
¼ engineer's time, 37 hrs., 37½c. 3.47
Total cost of labor $40.47
Cost of labor per ton 0.319
Steam required to operate fan engine and feed pump, per hr. at 130 pounds gauge pressure ..... 694 lb.
Equivalent from and at 212° per hr. ..... 843 lb.
Steam consumption per ton ..... 219 lb.
Value of steam consumed per ton of refuse incinerated at contract price of 4c. per 100 lbs. $0.088
Labor ..... .319
Total gross cost per ton $0.407
Revenue for steam per ton of refuse incinerated at contract price of 4c. per 100 lbs. for 2,680 lbs. of steam.....$1.072
Net profit per ton 0.665

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NOTES ON COMPUTATIONS IN BUILDING CONSTRUCTION IN NEW YORK CITY.*

By J. K. FINCH.

The following notes are not intended to cover completely the subject of computations in building construction in New York City but simply to give an idea of how some of these computations may be handled so as to save time and labor, and to explain the construction of the tables, etc., used in making such computations. They are based on the existing Building Code as adopted by the Board of Aldermen, September, 1899.†

This code divides buildings into three classes as regards construction: (1) Frame; (2) non-fireproof, and (3) fireproof buildings. The construction of frame buildings is limited to outlying districts by placing "fire limits" inside which no frame structures may be built. By frame buildings is meant those in which the exterior walls or a portion thereof are constructed of wood. This type of structure will not be dealt with although some of the formulæ given apply to such construction. The I. C. S. books on architecture give some good information on this subject. See also Kidder's "Building Construction," Part II.

The second class covers all buildings constructed within the fire limits which are not fireproof. The code states that all buildings used as hotels, schools, theaters, jails, police stations and hospitals over 35 feet (3 stories) in height (measured from center of curb to top of roof beams) and every other building over 75 feet (6 stories) must be fireproof. By fireproof is meant construction with walls of brick, stone, etc., floors of reinforced concrete or terra cotta blocks, fireproofed trim, etc.

In making computations for a building the kind of foundation necessary is one of the first things to decide on but one of the last things to be designed. It is necessary to know how the weight of

* Paper presented before the Columbia University Civil Engineering Society.

† Copies of this Code with amendments to date may be obtained from the Bureau of Buildings, 18th St. and 4th Ave., N. Y.

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the building is to be taken care of as regards foundations so that its construction as regards layout of walls, columns, etc., may be allowed for, but before the foundations can be designed in detail we must know the weights coming on them. The question of layout of columns, etc., with such questions as whether to use self-supporting or supported enclosure walls for skeleton steel construction, etc., would lead to the discussion of what might properly be called the "Economics of Building Construction," a subject which will not be taken up in these notes. Engineers are often called upon to place the structural details in such position that they will not interfere with the architect's designs and floor plans and there should always be the fullest coöperation at this point, as slight changes may often prove profitable.

The scheme for computations for a building will be taken up as follows: (1) floor construction, (2) roof construction, (3) girders and columns, (4) walls, and (5) foundations.

Since a large amount of the design of beams and girders is most easily done by means of a constant it would seem best to explain this method before taking up any special cases.

Designing Beams by Means of a Constant.

If we let $C =$ a constant, sometimes called the coefficient of strength of a beam and put

$$C = 8M \tag{1}$$

where C is the constant in ft. lbs. and M is the bending moment in ft. lbs., then, from the usual formula, M in in. lbs. $= kI/d$ or M in ft. lbs.

$$= \frac{kI}{12d} = \frac{kS}{12} \quad \text{and} \quad C = \frac{2}{3}kS \tag{2}$$

where

$k =$ allowable extreme fiber stress in lbs. per sq. in.,

$I =$ moment of inertia of beam section in ins.^{4},

$d =$ distance of extreme fiber from neutral axis in ins., and

$S =$ section modulus in ins.$^3 = \frac{I}{d}$,

and for a uniformly loaded beam

$$C = 8M = 8\left(\frac{wl^2}{8}\right) = wl^2 = Wl \tag{3}$$

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where

w = uniform load in lbs. per lin. ft.,
l = span of beam in feet, and
W = total uniform load in lbs.

Hence, knowing the value of C for any beam, we may obtain the safe uniformly distributed load for that beam by simply dividing by the span in feet and the safe load in lbs. per lin. ft. by dividing this by the span. For any form of loading other than uniform loads we determine M in ft. lbs. by the usual methods and multiply by 8 which gives the value of C necessary to support this load.

As values of C are given in the steel handbooks for various beams and are easily obtained from formula 2 and tabulated for other materials this method of figuring necessary size of beams or girders saves much time and labor.

Computations for Non-Fireproof Buildings.

Residences, small loft buildings, etc.; Brick, stone, etc., walls; wood floors and trim.

1. *Floor Construction.*

The live load it is necessary to design for is specified in the code and varies from 60 lbs. per sq. ft. for residences to 150 for factories. The dead load depends on whether single or double flooring is used, kind of ceiling used, etc. See Table I. for weights. The bending moments are found in the usual way, allowing for the live load, dead load, partitions resting on floor, ceilings, etc.

Necessary thickness for plank, beams, etc., is determined by the following formula given in the code:

$$W = \frac{bd^2}{l} k \qquad (4)$$

where

W = safe total load in lbs.,
b = breadth of beam in ins.,
d = depth of beam in ins.,
l = span of beam in ft.,
k = a constant depending on kind of wood
= 70 for hemlock,
90 for spruce and white pine,
120 for oak,
140 for yellow pine.

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Since from (3) we have $C = Wl$ from (4) we obtain

$$C = bd^2k \tag{5}$$

From formulæ (4) and (5) we may construct tables for plank and beams of different woods similar to Tables II. and III. for yellow pine given herewith. The plank table is most easily used in the form given—safe loads in lbs. per sq. ft. for different spans and thickness of plank. Use (4) for this changing l to inches. Table III. is easily constructed from (5) since for any depth of beam, 1 inch wide, $C = d^2k$ and for any other width we simply multiply by the width in ins.

When the ceilings are plastered on lath fastened directly to the underside of the beams it is necessary to limit the deflection of the beam so the plaster will not crack. This deflection limit is given as one 360th of the span or $\frac{1}{30}$ inch per foot.

Since

$$d'' = 22.5\frac{wl^4}{EI} \tag{6}$$

the usual formula for deflection of a uniformly loaded beam where

$d'' =$ deflection in ins.,
$w =$ uniform load in lbs. per lin. ft., etc.

If we let D equal the constant to be used so that the deflection limit shall not be exceeded $= wl^2$, then from (6)

$$D = \frac{EId''}{22.5l^2}$$

or since $d'' = \frac{l}{30}$ we have

$$D = \frac{EI}{675l}$$

substituting for $E = 2{,}000{,}000$ lbs. per sq. in. and for a rectangular beam $I = bd^3/12$ we get

$$D = 247\frac{bd^3}{l} \tag{7}$$

or if we let $D = wl^3 = 8Ml$ then $D = 247bd^3$.

See Table IV.

This formula must be used to determine the allowable load for

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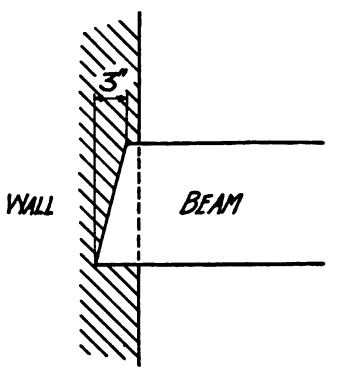
beams over a certain length so the deflection limit will not be exceeded. Using (5) for yellow pine and (7) as above we have

$$140bd^2 = 247\frac{bd^3}{l} \quad \text{or} \quad l = 1\tfrac{3}{4}d.$$

That is, we may use formula (5) for spans up to $1\frac{3}{4}\ d$ (l in ft., d in ins.) for *Y. P.* without exceeding the limit of deflection, over that we must use (7). Similarly for oak, spruce, etc., we may obtain the limits.

The above formula cannot be used for very short spans for in such cases the allowable shear may be exceeded. See Table V.

The bearing necessary for ends of beams is easily determined knowing the width of beam, reaction and allowable compression from Table V. The ends of all beams resting in walls are beveled 3" so in case of fire they will fall clear without injuring the wall.



2. *Roof Construction.*

The live load specified in the code is 50 lbs. per sq. ft. up to 20° pitch and 30 lbs. for over that, measured on a horizontal plane. The dead load depends on the kind of roof. See Table I.

3. *Girders and Columns.*

For details of construction see Kidder's "Building Construction and Superintendence," Part II. Girders are designed by the same formulæ given above. The formulæ for columns are as follows:

Long leaf pine,

$$p = 1{,}000 - 18\frac{l}{d} \tag{8}$$

White pine and spruce,

$$p = 800 - 15\frac{l}{d} \tag{9}$$

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Oak,

$$p = 900 - 17\frac{l}{d} \qquad (10)$$

Chestnut and hemlock, ⅝ white pine,
Locust, 1½ white pine,

where

p = safe load in lbs. per sq. in.,
l = length in ins.,
d = least diameter in ins.

The limit of length is $\frac{l}{d} = 30$.

See Table IV. for values for compression, etc., which make necessary the use of caps, base plates, etc., to carry the load from one column to another.

In computing the load coming on columns the code allows a reduction in the live load on the ground that the floors will never be entirely loaded.

4. *Walls.*

The thickness is specified in the code for different heights, etc. The allowable pressure on brickwork is given in Table V. and where beams, etc., rest on the walls they must be increased in thickness by buttresses if this pressure is exceeded. Isolated brick piers are also limited in size by the allowable pressure on brickwork and cannot exceed in height 10 times their least dimension.

The following formula from Kidder is useful in figuring piers:

$$p = P - 6\frac{H}{D} \qquad (11)$$

where

p = allowable unit stress in lbs. per sq. in.,
P = allowable unit stress in lbs. for direct compression,
H = height of pier in ins.,
D = least diameter of pier in ins.

Lintels in brick and stone walls are of stone or iron, wood is not allowed except for inside lintels. The load coming on the lintel is usually taken as that part of the wall in a 60° triangle. See Table V. for values of bending for stone, cast iron, steel, etc. Steel beams are most commonly used for lintels. Table VI. giving the constants for channels used as lintels is useful. Similar tables can be constructed for angles, etc., from formula (12).

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5. *Foundations.*

The minimum size of footings, allowable pressure on different soils and on piles, etc., are given in the code. Taking the dead load of walls, roof, and floors and the live load on roof and floors (decreased as allowed for in the code) it is a simple matter to figure the load per lin. ft. on wall footings and, knowing the minimum width allowed, see if it is necessary to increase this width in order to keep the footing pressure under that allowed for the material the footing is on. Footings for columns are handled in the same way.

Computations for Fireproof Buildings.

There are two forms of construction used. Reinforced concrete and steel. The cost of the former is so great, due to the restrictions of the code as regards concrete construction, that there has been very little of this work done in New York in comparison with other cities. We will, therefore, confine ourselves to the latter form—steel columns, girders and floor beams with fireproof arches and brick or stone enclosure walls.

1. *Floor Construction.*—There are three forms of fireproof floor construction: (1) Reinforced concrete, (2) terra cotta arches, (3) brick. Brick arches are used at present only for very heavy construction such as boiler room floors, sidewalk arches, etc., the code requires a rise of 1¾ ins. or over for each ft. of span and a thickness of 4 ins. for spans 5 ft. or less and 8 ins. for over 5 ft. As regards 1 and 2 there are many points in favor of each type and it is an open question which is the better. Reinforced concrete is cheaper (about 18 cents a sq. ft. of floor compared to 25 for terra cotta) but is heavier than terra cotta, thereby requiring more steel to support it. At present prices there is no doubt that concrete costs far less but as regards fireproofing qualities it is probable that it is less efficient.

There are three forms of floor arches used—flat, panel and arched. The latter is used for very heavy floor loads. There are also two types of construction—one which protects the bottoms and sides of the beam with a fireproof coating and one which does not. The wood floor is laid on wood sleepers usually 3″ × 3″ and 16 ins. on centers. These sleepers are laid on top of the iron floor beams and the arches must be made up even with the top of the

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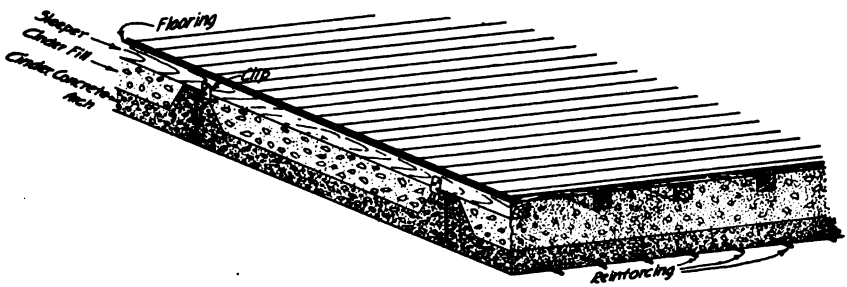
beams with cinder concrete. The space between the sleepers is filled in with lean cinder concrete and the sleepers are held to the beams by means of steel clips (see Fig. 1). The ceilings are



SECTION OF FIRE PROOF FLOOR
SHOWING CONSTRUCTION

FIG. 1.

usually plastered directly on the underside of the floor arches but hanging ceilings are sometimes used. The spans usually used are from 5 to 10 feet and for any building the most economic span is determined by trial, remembering that decreasing span decreases cost of arch but increases weight of steel, etc.

*Reinforced Concrete Arches.**—The question of the amount of reinforcing steel and thickness of arch is not subject to computation but is fixed for different spans and systems by standards in the Building Department which have been approved after undergoing required strength and fire tests. Two kinds of reinforcing are used—bars and mesh. The mesh, or wire cloth, is cheaper when there are large areas to be covered and it can be rolled out over the beams with little cutting. The bars are easier to handle in small work where there is lots of fitting necessary but are open to the serious objection that the contractor can easily cheat on the reinforcing by putting the bars further apart on centers than called for—something it is impossible to do with cloth. Fig. 2 shows the forms of construction for the various concrete arches. Flat bottom concrete floor construction with double flooring, 9" beams and a 4" arch would weigh complete about as follows:

* See catalogue of Adin G. Pierce Co. for useful information, etc.

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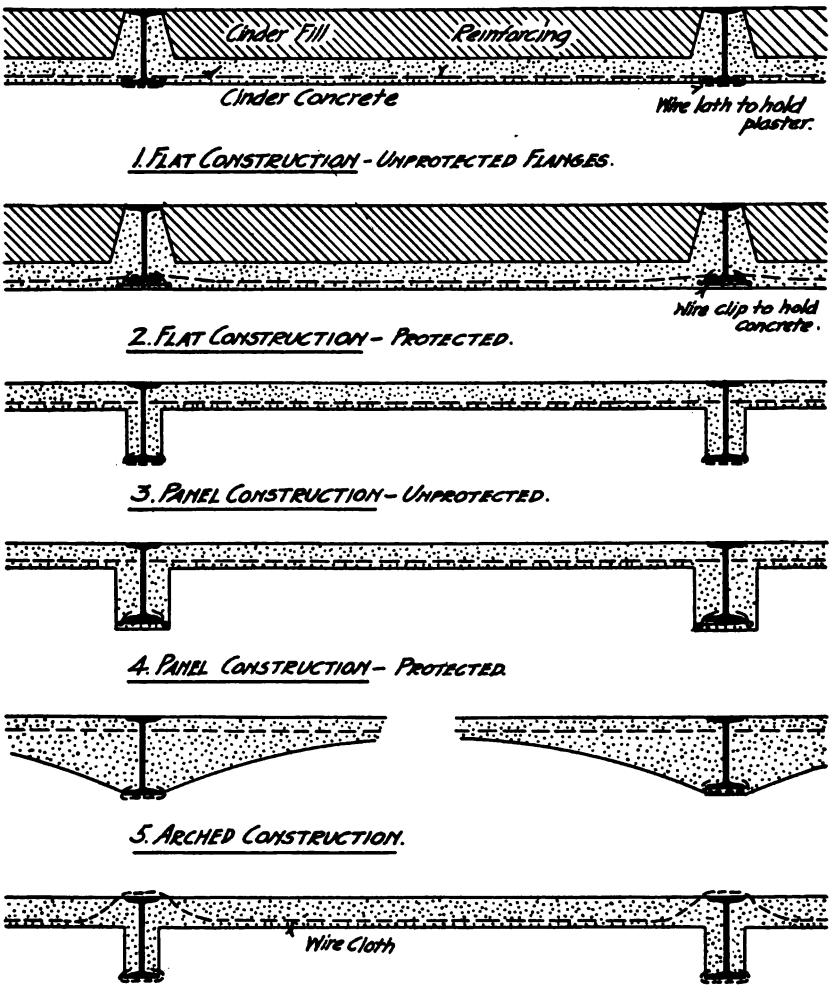

FIG. 2.

	Lbs. per sq. ft.
Plaster, 2 coats	= 6
4 inch concrete arch	= 26
5 inch cinder fill	= 15
3 inch sleeper fill including sleepers	= 15
Under flooring	= 2
Finished flooring	= 3
Total	67

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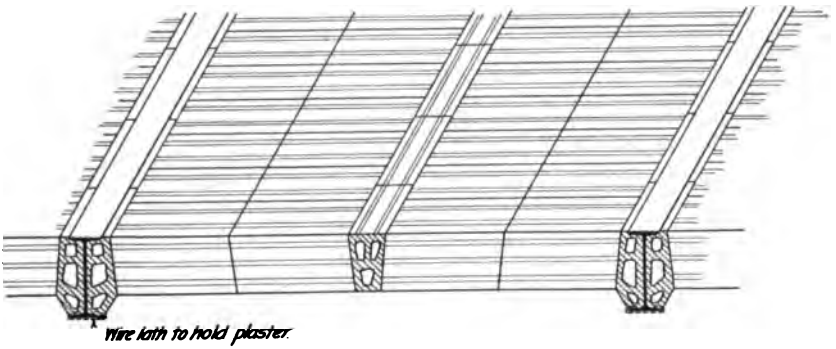
*Terra Cotta Arches.**—There are two methods of construction used for these arches—side and end. (See Fig. 3.) The end construction is much stronger (50 per cent.) than the side but



END CONSTRUCTION – PANEL ARCH, UNPROTECTED FLANGES.

SIDE CONSTRUCTION – FLAT ARCH WITH PROTECTED FLANGES.

FIG. 3.

requires more skilful and careful labor to erect it properly. The code requires a depth of not less than 1¾ ins. for each foot of span for flat arches and a rise of not less than 1¼ ins. for each foot and minimum thickness of 6 ins. for arched construction.

The following table gives the weight, etc., for side and end construction and also the *C* in ft. lbs. for a factor of safety of 7 from tests.

* See catalogue of National Fireproofing Co. for useful information, etc.

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Depth of Arch, Inches.	Weight, Lbs. Per Square Foot.	Allowable Span.	C for Side Construction.	C for End Construction.
6	26	3′ 5″	3,200	4,350
7	27	4′ 0″	3,700	5,550
8	31	4′ 7″	4,200	6,900
9	35	5′ 1″	4,750	8,400
10	37	5′ 8″	7,000	10,050
12	42	6′10″	8,175	13,700

Design of Steel Beams.—The steel handbooks tabulate the values of C for the various sections. The Cambria handbook gives the values under the heading of F and F' where $F = C$ for a fiber stress of 16,000 lbs. per sq. in. as allowed by the code and $F' = C$ for 12,500 lbs. per sq. in. for use in bridge design. These values are obtained from formula (2) which becomes

$$C = 10{,}667\,S \qquad (12)$$

when we substitute for K 16,000 lbs. per sq. in.

Approximate Rule.—For standard sections C in ton lbs. may be obtained within about 5 per cent. by multiplying the weight of the beam in lbs. per lin. ft. by half its depth. Thus for the standard 12″ I @ 31½.

$$C = 6 \times 31\tfrac{1}{2} = 189 \text{ ton lbs.},$$
$$= 189 \times 2000 = 378{,}000 \text{ ft. lbs.}$$

(The actual value from Cambria is 383,670.) On a span of 10 ft. this beam would hold approximately 37,800 lbs. or 3,780 lbs. per lin. ft.

Standard Sections.—Remember that this approximate rule only holds for standard sections. These are the sections which can always be obtained and are usually printed first for each size of beam in the handbooks. Thus 12 I 31½ is standard but 12 I 40 is special and as specials are not rolled at the mills as frequently as standard we should always use the standard section unless we need a large amount of one size and time is not important. As a matter of fact we can fit almost any condition with little waste of material by selecting between a standard I or [, by modifying the span, etc.

Deflection.—The code limits the deflection of a steel beam to $\frac{1}{30}$ inch per foot which is the limit for plastered ceilings.

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Since

$$d'' = 22.5\frac{wl4}{EI} = \frac{l}{30}$$

then

$$l = \frac{EI}{675C} = \frac{EI}{675 \times 10{,}667S} = \frac{Ed}{675 \times 10{,}667} = 4d \qquad (13)$$

taking $E = 29{,}000{,}000$ and where $d =$ dist. from neutral axis to extreme fiber in ins. and $l =$ limit of span in ft. For a 12″ I the limit of span is therefore 24 ft. As it is not economical to use beams of such depth that the limit of span is exceeded and we cannot use 16,000 lbs. per sq. in. intensity of stress, we simply use beams of a depth which will allow us to obtain the full value of the beam.

Beam Tables.—All the steel handbooks give tables for the safe uniform load for various beams and different spans. Also for spacing of beams for different spans and a uniform load of 100 lbs. to the sq. ft. These tables are very useful in getting up a plan for a floor system where it is necessary to try different spans, spacings, etc., to determine the most economical layout.

2. *Roof Construction.*

The live load has already been given in the notes on wood beams. The dead load is easily figured from Table I. and the values given above for floor construction.

3. *Girders and Columns.*

Girders are figured by the familiar $M = KI/d$ formula. Columns are either of cast iron, wrought iron or steel. The code gives:*

Cast iron, $p = 11{,}300 - 30\,l/r$ (14)

Wrought iron, $p = 14{,}000 - 80\,l/r$ (15)

Steel, $p = 15{,}200 - 58\,l/r$ (16)

when

$p =$ allowable stress in lbs. per sq. in.,

$l =$ length of column in ins., and

$r =$ least radius of gyration in ins.

* Neither these formulæ nor those for wood columns are given in the code in this form, but these formulæ may be obtained by working backward from the values given in the code.

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Columns			1 & 4		2 & 3		5-6-7 & 13		8 & 10		9	
Story Heights	R		Tons		Tons		Tons		Tons		Tons	
11'4"	12	D.L.	3 0		10 5		14 0	Above Tank.	7 0		14 0	
		L.L.	1 4		5 0		6 5		3 2		6 5	
		W.L.	4 5		13 0				2 0		5 5	
		T.L.	8 9		28 5		20 5		13 0		26 0	
12'1"	11	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	3 8		11 7		15 0		7 5		15 0	
		W.L.	1 5		6 0				5 0		10 0	
		T.L.	15 4		52 1		51 0		29 5		58 5	
12'1"	10	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	3 8		11 1		14 2		7 1		14 2	
		W.L.	10 0		30 0				5 0		14 0	
		T.L.	30 5		99 1		78 7		45 2		90 2	
12'1"	9	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	3 8		10 5		13 5		6 6		13 5	
		W.L.	6 5		14 5				5 0		10 0	
		T.L.	41 7		130 5		83 7		60 6		121 2	
12'1"	8	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	2 8		10 0		12 0		6 4		12 0	
		W.L.	6 0		9 0				5 0		10 0	
		T.L.	52 5		156 5		114 0		76 0		151 5	
12'1"	7	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	2 6		9 4		12 0		6 0		12 0	
		W.L.	6 0		9 0				5 0		10 0	
		T.L.	62 5		179 2		133 5		90 0		181 0	
12'2"	6	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	2 5		8 0		11 2		5 6		11 2	
		W.L.	6 0		9 0				5 0		10 0	
		T.L.	72 7		202 9		152 2		105 6		208 7	
12'2"	5	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	2 5		8 0		10 5		5 5		10 5	
		W.L.	6 0		9 0				6 0		15 5	
		T.L.	82 7		226 9		170 2		121 0		241 2	
12'2"	4	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	2 1		7 6		9 0		4 9		9 0	
		W.L.	6 0		9 0				6 0		15 5	
		T.L.	92 5		248 5		187 5		136 5		277 0	
12'2"	3	D.L.	1 7		5 9		7 5		5 0		7 5	
		L.L.	2 0		7 0		9 0		4 5		9 0	
		W.L.	6 5		10 0				6 0		15 5	
		T.L.	102 7		271 4		204 0		151 6		302 0	
13'6"	2	D.L.	2 2		7 0		10 0		5 0		10 0	
		L.L.	1 0		6 4		0 2		4 1		0 2	
		W.L.	11 0		20 0				6 0		15 5	
		T.L.	117 7		313 6		222 2		167 5		338 7	
15'6"	1	D.L.	2 2		7 0		10 0		5 5		11 0	
		L.L.	1 7		5 9		7 5		4 6		9 0	
		W.L.	5 0		6 0				7 5		15 0	
		T.L.	126 6		353 5		239 7		185 1		360 7	
12'0"	B	D.L.	3 5		10 0		7 5		5 0		10 5	
		L.L.	7 0		18 0		7 5		6 0		12 0	
		W.L.	5 0		2 0				13 4		8 3	
		T.L.	142 1		363 5		254 7		209 5		399 5	
Distance to bottom of base plates below cellar floor.			← Finished Cellar Floor → 8"± 1'4"± Col 1 Col 4.		8"± 1'4"± Col 2. Col 3		4'4" Cols 5-6-13	8"± Col. 6&7.	11"±		8"±	

Assumed Load—

Live Load - All Floors. 120 lbs. per sq. ft. Roof 50 lbs. per sq. ft.
Dead " - 2nd & 3rd " 80 " " " " Other Floors 60 lbs. per sq. ft. Roof 110 lbs. per sq. ft.

Sections have been increased as necessary to allow for eccentric loading.

Column Schedule

Building to be erected at Nos. 251-252 West 117th St. N.Y. City.

Set (E) Sheet (12)

Fig. 4.

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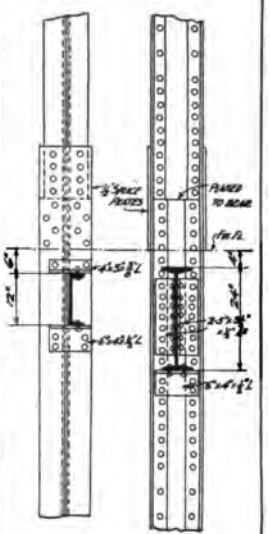
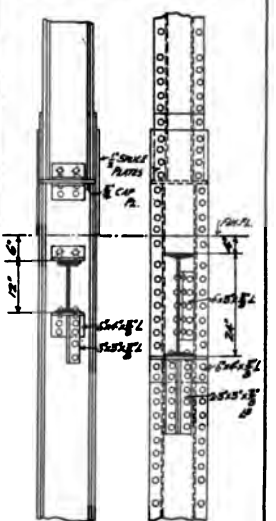
11 & 12			14.			15 & 16			17.		
Tons		Str. Tanks.	Tons.			Tons		Str. Tanks.	Tons		
20 0						20 0					
14 0			27 5			27 5					
6 5		Lattice	12 5		Lattice	12 5		Lattice			
40 5			40 0			60 0					
7 5			15 0			15 0					
15 0			30 0			30 0					
63 0			85 0			105 0					
7 5			15 0			15 0					
14 2			20 5			20 5					
84 7			120 5			140 5					
7 5			15 0			15 0					
13 5			27 0			27 0					
105 7			170 5			190 5					
7 5			15 0			15 0					
12 8			25 5			25 5					
126 0			211 0			231 0					
7 5			15 0			15 0					
12 0			24 0			24 0					
145 5			250 0			270 0					
7 5			15 0			15 0					
11 2			22 5			22 5					
164 2			287 5			307 5					
7 5			15 0			15 0					
10 5			21 0			21 0					
182 2			323 5			343 5					
7 5			15 0			15 0					
9 8			19 5			19 5					
199 5			358 0			378 0					
7 5			15 0			15 0					
9 0			18 0			18 0					
216 0			391 0			411 0					
10 0			20 0			20 0					
8 2			16 5			16 5					
234 2			427 5			447 5					
10 0			20 0			20 0					
7 5			15 0			15 0					
251 7			462 5			482 5					
7 5			15 0			15 0			9 0		
7 5			15 0			15 0			24 0		
266 7			492 5			512 5			33 0		
8'			8'			8'			8'		



-Typical P. & L. Column-



-Typical P. & L. Column-

Smith and Jones, Archts.
26 Broadway. New York.

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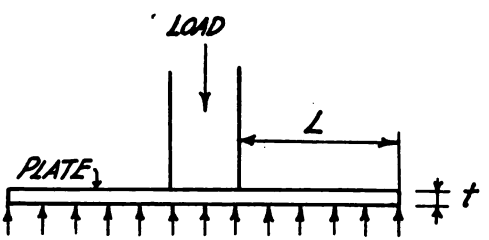
For computing columns architects have tables giving r for various built up sections and it is well to tabulate the values of r we work out for different sections for future use. The new Bethlehem H sections are being largely used for columns at present as they are more economical, requiring practically no fabrication. Values of r for these sections are given in the Bethlehem Steel Handbook.

In computing the loads coming on columns it is usual to construct a "column schedule." This is merely a tabulation of loads and statement of sizes, etc., necessary. Fig. 4 is a column schedule for a 12-story loft building. The horizontal divisions represent the various floors as marked in the second column—B standing for basement and R for roof. The first column gives the heights of the floors—finish to finish. The numbers in the top horizontal section represent the numbers of the columns as given on the plans. Thus columns Nos. 1 and 4 receive the same load and are alike as regards make up. D.L. stands for dead load, L.L. live load, W.L. wall load (walls resting on floor beams or girders), T.L. total load for that floor. The loads are given in tons and tenths. As is usual a typical column detail is shown and it will also be seen that the columns are built in two story lengths except the top section which is three stories. The assumed loads are also given and the sections have been increased for eccentric loading by changing the eccentric load to equivalent central loading, it being assumed that an eccentric load is uniformly distributed over the section at the next floor level below.

4. *Walls.*

The thickness, etc., is fixed by the code depending whether the wall is a bearing wall, self-supporting curtain (or enclosure) wall



or a supported wall. Where beams rest on a wall it is necessary to use bearing plates to distribute the load over a large enough

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area so that the allowable compression on the wall is not exceeded. The following formula is useful in determining required thickness for bearing plates.

The plate may be taken as a uniformly loaded cantilever in which case $M = wL^2/2$

where w = uiform load on plate = allowable pressure on wall or footing in lbs. per sq. in.,

L = length of projection in ins.

Since $M = kS = kt^2/6$ for a section 1" wide where t = thickness in ins. we have

$$\frac{wL^2}{2} = \frac{kt^2}{6} \quad \text{or} \quad t = \sqrt{\frac{3wL^2}{k}} = L\sqrt{\frac{3w}{k}} \tag{17}$$

for a steel plate or a brick wall laid in cement $k = 16{,}000$ and $w = 250$ or

$$t = 0.217\,L \tag{18}$$

and similarly for granite, bluestone and other bearing plates on various kinds of walls.

The length of a beam necessary for bearing depends on its depth and the thickness of the web. If the length of bearing is not great enough the beam will fail by buckling or crippling of the web. As the intensity of shear is greatest along a section at 45° to the neutral axis we may consider the web as a series of columns of length $C\sqrt{2}$ where C is the clear depth of web in ins. between flanges. If in formula (16) we substitute for l, $C\sqrt{2}$ and for r put its value $0.289t$ where t is the thickness of the web we get

$$p = 15{,}200 - 284\,C/t. \tag{19}$$

This value is usually about 8,000 lbs. per sq. in. for I beams. The necessary length of bearing in ins. is

$$\frac{R}{p \cdot t} \tag{20}$$

where R is the reaction or load in lbs.

5. *Foundations.*

The question of computations and designs for foundations is so fully taken up in the work at college it will not be discussed here.

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Miscellaneous Formulæ.

The following formulæ collected from various sources may be easily established from experimental data, represent safe practice and are in such convenient form that they are here given with the hope that they may prove useful although they are not directly connected with building construction.

Rope.

Six strands, 19 wires each, hemp center.

Iron hoisting rope,

$$P = 680\,c^2.$$

Steel hoisting rope,

$$P = 1{,}500\,c^2.$$

Manilla or hemp rope,

$$P = 100\,c^2.$$

Where P = safe working load in lbs., C = circumference of rope in ins. obtained by measuring with a tape (= about 2.8 times greatest diameter).

Chain.

$$P = 12{,}600\,d^2$$

where d = the diameter in ins. of the iron rod the links are made of.

*Earth Pressure.**

$$P = hX/3$$

where P = pressure in lbs. per sq. ft. at depth h in ft. and X = wt. of earth in lbs. per cu. ft.

Decimals of Inches to 16ths.

Divide hundredths by 6. Thus .67 in. $= 67 \div 6 = \frac{11}{16}$.

Concrete. (Fuller's Rule.)

To obtain the approximate amounts of cement, sand and broken stone in one cubic yard of average broken stone concrete. Divide 11 by the sum of the parts, equals the number of barrels of cement. Multiply this by the number of parts of sand and by the number of cubic feet in a barrel of cement (3.8 average) and divide by 27 equals the number of cubic yards of sand required. Similarly for the broken stone.†

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* Cain's formula.

† See "The Laws of Proportioning Concrete," *Trans. Am. Soc. C. E.*, Vol. LIX., p. 67, 1907.

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Example:

Find the quantities for one yard of 1, 3, 5 Portland cement concrete.

$$\frac{11}{9} = 1.22 \text{ bbls. of cement to the yard.}$$

$$\frac{1.2 \times 3 \times 3.8}{27} = .51 \text{ cu. yds. sand.}$$

$$\frac{1.2 \times 5 \times 3.8}{27} = .84 \text{ cu. yds. broken stone.}$$

TABLE I.

WEIGHTS OF MATERIALS USED IN BUILDING CONSTRUCTION.

Brickwork (Code)	115	lbs. per cu. ft.
Granite	170	" " " "
Limestone	165	" " " "
Concrete (stone)	165	" " " "
Mortar	100	" " " "
Cinder concrete	80	" " " "
Cinder fill	36	" " " "
Steel	490	" " " "
Wrought iron	480	" " " "
Cast iron	450	" " " "
Wood	2 to 4	" " bd. ft.
Plaster (2 coats)	5	" " sq. ft.
Plaster (3 coats metal lath)	10	" " " "
Galvanized iron roofing	1 to 4	" " " "
Slate roofing	6	" " " "
Tin roofing	1	" " " "
Tar roofing	2 to 5	" " " "
T. C. partitions 3"	16	" " " "
T. C. partitions 4"	17	" " " "
T. C. partitions 6"	25	" " " "

TABLE II.

SAFE LOADS IN LBS. PER SQ. FT. FOR YELLOW PINE PLANK OF VARIOUS THICKNESSES WITH DIFFERENT SPANS.

$w = 20{,}160\ bd^2/l^2$ where l is in ins.

Span in Ins.	Thickness of Plank in Ins.					
	⅞	1	1⅛	1¼	2	3
12	107	140	176	219	560	—
14	79	103	130	161	412	—
16	—	79	100	123	316	—
20	—	—	64	79	201	452
24	—	—	—	—	140	315
36	—	—	—	—	62	140

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TABLE III.

CONSTANT C IN FT. LBS. FOR YELLOW PINE BEAMS.

$C = 8\ M = wl^2 = 140\ bd^2$.

Depth in Inches.	Thickness in Inches.				Max. Span for Limit Defl. $= 1\frac{1}{4}$ d.
	1	3	4	6	
4	2,240	6,720	8,960	13,440	7′ 0″
6	5,040	15,120	20,160	30,240	10′ 2″
8	8,960	26,880	35,840	53,760	14′ 0″
10	14,000	42,000	56,000	84,000	17′ 6″
12	20,160	60,480	80,640	120,960	21′ 0″
14	27,440	82,320	109,760	164,640	24′ 6″
16	35,840	107,520	143,360	215,040	28′ 0″
18	45,360	136,080	181,440	272,160	31′ 6″

TABLE IV.

CONSTANT D IN FT. LBS. FOR WOOD BEAMS.

$D = 8\ Ml = wl^2 = 247\ bd^2$.

Depth in Ins.	Thickness in Ins.			
	1	3	4	6
4	15,800	47,400	63,200	94,800
6	53,350	160,050	213,400	320,100
8	126,400	379,200	505,600	758,400
10	247,000	741,000	988,000	1,482,000
12	427,000	1,281,000	1,708,000	2,562,000
14	678,000	2,034,000	2,712,000	4,068,000
16	1,012,000	3,036,000	4,048,000	6,072,000
18	1,440,000	4,320,000	5,760,000	8,640,000

TABLE V.

ALLOWABLE STRESSES IN LBS. PER SQ. IN. AS GIVEN IN THE NEW YORK BUILDING CODE.

Compression.

Steel	16,000
Wrought iron	12,000
Cast iron	16,000

	With Grain.	Across Grain.
Oak	900	800
Yellow pine	1,000	600
White pine	800	400
Spruce	800	400
Locust	1,200	1,000
Hemlock	500	500
Chestnut	500	1,000

P. C. Concrete 1, 2, 4	230
P. C. Concrete 1, 2, 5	208
Rubble stonework P. C.	140
Rubble stonework Rosendale.	111
Rubble stonework Lime	70
Brickwork P. C.	250
Brickwork Rosendale	208
Brickwork Lime	111
Bluestone	2,000
Slate	1,000
Granite	1,000 to 2,400
Marble	600 to 1,200

Tension.

Steel	16,000
Wrought iron	12,000
Cast iron	3,000
Yellow pine	1,200
Oak	1,000
White pine and spruce	800
Hemlock	600

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Shear.	
Steel web plates	9,000
Steel shop rivets and pins.	10,000
Steel fired rivets	8,000
Steel fired bolts	7,000
Wrought iron bolts, 6,000, 7,500, 6,000, and	5,500
Cast iron	3,000

	With Fibre.	Across Fibre.
Yellow pine	70	500
White pine	40	250
Spruce	50	320
Oak	100	600
Locust	100	720
Hemlock	40	275
Chestnut	—	150

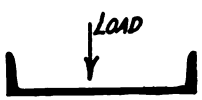
SAFE EXTREME FIBER STRESS FOR BENDING.

Rolled steel beams	16,000
Riveted steel beams	14,000
Rolled wrought iron beams	12,000
Riveted wrought iron beams	12,000
Cast iron (compression side)	16,000
Cast iron (tension side)	3,000
Woods—as in tension.	
Granite	180
Gneiss (New York City)	150
Limestone	150
Slate	400
Marble	120
Sandstone	100
Bluestone (North River)	300
Concrete P. C. 1, 2, 4	30
Concrete P. C. 1, 2, 5	20
Brickwork in cement	30

TABLE VI.

CONSTANTS FOR I BEAMS USED AS LINTELS.

$C = 10{,}667\ S$, max. span $= 4(b - x)$.



Size.	Weight, lbs.	C in ft. lbs.	Max. Span for Limit Defl.
3″	4.00	2,200	3′ 9″
4	5.25	3,100	4′ 5″
5	6.50	4,100	5′ 0″
6	8.00	5,300	5′ 6″
7	9.75	6,700	5′ 11″
8	11.25	8,400	6′ 8″
9	13.25	10,400	7′ 3″
10	15.00	12,500	7′ 11″
12	20.50	18,700	9′ 0″
15	33.00	33,700	10′ 6″

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UNIVERSITY NOTES.

DEPARTMENT OF CHEMISTRY.

(From the annual report of the head of the department for the year 1909–10.)

The changes in the teaching staff during the past year were as follows:

Charles Frederick Chandler, M.D., Ph.D., LL.D., Sc.D. (Oxon.), Mitchill professor of chemistry, has resigned after 54 years of active work, 46 of which were spent in teaching at Columbia University. During that period Professor Chandler has taught general, organic, and analytical chemistry and assaying, and for two years taught geology, beside having discharged from 1864–1897 the duties of dean, registrar, bursar and administrative head of the Department of Chemistry. In addition to his duties at the School of Mines, Professor Chandler was professor of chemistry and medical jurisprudence in the College of Physicians and Surgeons from 1873–1897, and has also been connected with the New York College of Pharmacy, having lectured there from 1866–1910. From 1894–1900 Dr. Chandler was vice-president of the latter institution; from 1900–1904, president; and from 1904–1910, vice-president. He has been granted sabbatical leave of absence for the year 1910–11.

Charles E. Pellew, E.M., adjunct professor of chemistry, has resigned and has been granted sabbatical leave of absence for the year 1910–11.

James S. C. Wells, Ph.D., adjunct professor of analytical chemistry (qualitative), resigned July 1, 1909.

Marston Taylor Bogert, LL.D., has been appointed administrative head of the Department of Chemistry for the year 1910–11.

Milton C. Whitaker, M.Sc., has been appointed professor of industrial and engineering chemistry.

Hal T. Beans, Ph.D., has been appointed instructor in analytical chemistry (qual.) and has been in charge of the qualitative laboratory for the past two years.

J. E. Zanetti, Ph.D., and W. C. Moore, A.M., have been appointed instructors in analytical chemistry (qual.).

Doris Whipple, B.S.; Charles McKone, A.M.; Robert F. MacCracken, B.S.; E. M. Flint, B.S.; E. C. Thompson, B.S.; B. G. Feinberg, B.S.; and F. L. Jouard, Ph.D., have served as assistants during a whole or a part of the academic year.

Summer School.—Professor J. Livingston R. Morgan will continue in charge of the summer school in chemistry during the coming summer (1910).

Attendance.—The details of attendance in the Department of Chemistry for the year 1909–10, are shown in the following tables:

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NUMBER OF DIFFERENT STUDENTS TAKING ONE OR MORE COURSES IN CHEMISTRY, 1909–10.

Applied Science, 1910	35	
Applied Science, 1911	60	
Applied Science, 1912	189	
Applied Science, 1913	227	
Applied Science, Special	13	524
Columbia College, 1910	29	
Columbia College, 1911	39	
Columbia College, 1912	74	
Columbia College, 1913	108	
Columbia College, Special	10	260
Pure Science, A.M.	27	
Pure Science, Ph.D.	13	
Teachers College	8	
Teachers College Extension	9	57
Individual students—Total		841
Registrations—Total		2,057

Departmental Notes.—During the past winter the Radium Institute was founded, of which Professor Chandler was one of the charter members. He subsequently became president of the Institute, but resigned that Dr. Butler might be made president.

On March 25, Professor Chandler delivered a lecture on "The Plant" at New Bedford, Mass.

On March 28, a banquet was given in honor of Prof. Chandler at the Hotel Astor by the alumni of the College of Pharmacy.

On April 2, a testimonial dinner was tendered to Dr. Chandler by the alumni of Columbia University. The dinner was held at the Waldorf Astoria and was attended by over 500 alumni, graduates of all classes from 1847 to the present date. On this occasion the Charles Frederick Chandler Lectureship was founded. These lectures are to be given annually by some prominent chemist at Columbia University.

On April 30, Dr. Chandler was the guest of honor of the various chemical societies at a banquet at the Waldorf-Astoria, at which a bust of heroic size was presented to Dr. Chandler and at his request will be presented to Columbia University and placed in the corridor of Havemeyer Hall. A duplicate of this bust was presented to Mrs. Chandler and will be given by her to the Chemists Club.

On May 13, a Kommers was held in honor of Dr. Chandler by the students, officers and alumni of the university.

The trustees of Columbia University passed a resolution that the chemical museum in Havemeyer Hall be known as the Chandler Chemical Museum.

Organic Laboratory.—The organic laboratory work for students in chemical engineering has been transferred from the summer session to the regular year. A new course (Chemistry 151–2) has been established, designed particularly to meet the needs of those especially

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interested in the application of organic chemistry to the solution of problems in medicine, physiology, etc., and the synthesis and study of individual organic compounds of importance in these fields. Synthetic drugs, alkaloids, essential oils, sugars, fats, and other groups of special interest to the medical profession are considered.

Ross A. Gortner, Ph.D. (1909), is now chemist in charge of the biochemical investigations for the Carnegie Institution's station for experimental evolution at Cold Spring Harbor, N. Y.

Alfred H. Kropff, Ph.D. (1909), is acting as Professor Bogert's special research assistant.

For about a month during November and December last, Dr. Hoffman, assistant in organic chemistry, was by special arrangement also in charge of organic and physical chemistry at the Brooklyn Polytechnic Institute on account of the serious illness of Professor Fay of the latter institution. Dr. Hoffman lectured before the chemical department of Wesleyan University, April 9, 1910, on the manufacture of natural and artificial perfumes.

Dr. Nelson has been appointed reader in chemistry for the College Entrance Examination Board.

At the twentieth anniversary celebration at Clark University September 14–17, 1909, Professor Bogert presented papers on "The Quinazolines" and on "The Teaching of Organic Chemistry" in the chemical conferences and at the public session of September 16, delivered an address upon "Chemistry and Conservation" following which the degree of LL.D., *honoris causa,* was conferred upon him by President G. Stanley Hall.

During the past year Professor Bogert has lectured on "Chemistry and Conservation" to the New York Republican Club; on the "Conservation of our Natural Resources" to the Phi Beta Kappa Society of Rutgers College; and on "Coal Tar Products" before the Men's Club of the St. Nicholas Ave. Presbyterian Church.

The investigations now under way in the organic laboratory include studies in the following fields: Quinazolines, stilbazols, aromatic nitriles, benzimidazols, bruccine, quinolines, naphthotetrazines, terpenes, new groups of artificial dyestuffs, synthetic drugs, ketone condensations methyl iodide additions to nitrogen heterocycles and various problems in stereo-chemistry.

The experience of this department for the past few years has shown that the supply of competent teachers of chemistry is far short of the demand. Our Ph.D. men are usually placed some time before they are graduated and the opportunities for a good teaching position are very bright indeed for men with this degree from Columbia.

Physical Chemistry Laboratories.—Three graduate students, two candidates for M.A. and one for Ph.D., have been continuing the research work on "The Weight of a Falling Drop" started in this laboratory four year ago and five others are expected during the summer.

Owing to the large classes in chemical engineering the number of students receiving instruction is considerably larger this year than last, a total of 82 having registered.

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Industrial Chemistry.—In the autumn of 1909, Professor Pellew gave a series of four public lectures on the "Practical Application of Modern Dyestuffs" under the auspices of the School of Fine Arts of Columbia University.

Professor Pellew was invited to Bridgeport, Conn., to address the members of the Bridgeport Art League upon the relations of modern dyestuffs to handicraft work. The lecture was illustrated with a large assortment of specimens of textile work illustrating ancient and modern methods of dyeing.

On April 22, Professor Pellew lectured upon the same subject before the Teachers Association at Plainfield, N. J.

Mr. Samuel Baer, under Professor Pellew's direction, has made a study of the application of the aniline black processes for use in the art of stencilling, and worked out a very satisfactory formula for a black stencil paste which can be readily applied by hand, by stencilling, or by block printing, and after a simple fixing with the flat iron, will give excellent results fast to both light and washing.

The Lewis Publishing Co., of St. Louis, have arranged to publish in the autumn Professor Pellew's book on handicraft dyeing and also to sell the apparatus, dyestuffs, stencil pastes, etc., to be used in connection with the work.

Professor Pellew expects to devote his time in the future to developing this particular line of chemical work.

Organic Analysis and Food Chemistry.—Several problems in organic analysis and food chemistry have been under investigation during the year and others will follow during the summer as the result of the past year's work.

The department receives requests for many reprints of the paper published two years ago by Professor Sherman and Dr. Kropff in which was deduced a formula for calculating the calorific power of petroleum oils from their density, which is now employed by several of the largest producers and consumers of fuel oil.

Mr. E. C. Kendall, first incumbent of the Goldschmidt Fellowship, has continued the investigation of the amylases begun by Mr. Meyer under Professor Sherman's direction in 1907. The investigation is yielding most gratifying results and it is proposed to extend the work in several directions with special attention to the further purification of the amylases and the behavior of those of greatest economic importance. The demands upon this branch of the department for gratuitous public service in the form of public lectures, and expert work for civic organizations, such as the New York Milk Committee, and others interested in food chemistry, has been very great and has been met in so far as teaching duties of the staff would permit.

Electrochemical Laboratory.—This laboratory has been very busy during the past year and successful research work has been carried out along such lines as electrolytic preparation of magnesium and strontium, production of ferro-silicon, manufacture of tin nitride, a study of the production of aluminum, combination of nitrogen and strontium carbide, fixation of nitrogen by the low voltage arc, comparison of nickel chloride and sulphate for electroplating.

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New apparatus has been added as follows: Wanner, optical pyrometer, reading to 4,000° C.; General Electric vacuum furnace; Westinghouse precision watt-meter; oxy-acetylene welding outfit; apparatus for the use of nitrogen under pressure.

Quantitative Laboratory.—The number of students taking courses in quantitative analysis has been larger than ever before; in many cases two or more men were assigned to a single desk and in addition ten men were given desks in the organic laboratory. The number of students in the professional courses of chemistry and chemical engineering is double that of last year. The laboratory facilities are inadequate to meet the demands of the large number of students taking courses in this department. Several papers, the results of research work, will be published in the near future.

CONTRIBUTIONS FROM THE HAVEMEYER LABORATORIES, VOLUME XI. 1909–10. EDITED BY CHARLES F. CHANDLER.

No. 166. The Condensation of Acetone by Means of Calcium Oxide. By Alfred Hoffman. Journ. Am. Chem. Soc., *31*, 722 (June, 1909).

The Functions of Chemistry in the Conservation of our Natural Resources. By Marston Taylor Bogert. Journ. Am. Chem. Soc., *31*, 125 (February, 1909).

Chemistry and the Conservation of our Forests and Minerals. By Marston Taylor Bogert. School of Mines Quarterly, *30*, 217 (April, 1909).

VOLUME XII.

No. 167. On Some Amino and Nitroamino Derivatives of Benzoic, Metatoluic and Metaphthalic Acids. By Marston Taylor Bogert and Alfred H. Kropff. Journ. Am. Chem. Soc., *31*, 841 (July, 1909).

No. 168. Researches on Quinazolines (22d paper): On 2-Methyl-3-amino-4-quinazolone and Certain of its Derivatives. By Marston Taylor Bogert and Ross A. Gortner. Journ. Am. Chem. Soc., *31*, 943 (August, 1909).

No. 169. Researches on Quinazolines (23d paper): On 6-Methyl-7-amino-quinazolones, 7-Nitro-quinazolone-6-carboxylic Acids, and 1, 3, 7, 9-Naphtho-tetrazines. By Marston Taylor Bogert and Alfred H. Kropff. Journ. Am. Chem. Soc., *31*, 1071 (September, 1909).

No. 170. A New Rapid Volumetric Method for the Determination of Niobium in the Presence of Tantalum and its Application to the Analysis of Niobium Minerals. By F. J. Metzger and C. E. Taylor. School of Mines Quarterly, *30*, 323 (July, 1909).

No. 171. On the Nature of Certain Sodium-uranium Compounds. By F. J. Metzger and M. Heidelberger. Journ. Am. Chem. Soc., *31*, 1040 (September, 1909).

No. 172. A Study of the Effect of Excess of Reagent and Time of Reaction in the Determination of Iodine Numbers of Fatty Oils. By Dwight A. Bartlett and H. C. Sherman. School of Mines Quarterly, *31*, 55 (November, 1909).

No. 173. A Comparison of the Calculated and Determined Values for the Specific Temperature Reactions of Oil Mixtures with Sulphuric Acid. By W. H. Boynton and H. C. Sherman. School of Mines Quarterly, *31*, 64 (November, 1909).

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No. 174. Researches on Quinazolines (24th paper): On Oxalyl Anthranilic Compounds and Quinazolines Derived Therefrom. By Marston Taylor Bogert and Ross A. Gortner. Journ. Am. Chem. Soc., *32*, 119 (January, 1910).

No. 175. A Source of Error in the Examination of Foods for Salicylic Acid. By H. C. Sherman. Journ. of Ind. and Eng. Chem., *2*, 24 (January, 1910).

No. 176. The Determination of Cerium in Cerite and Monazite. By F. J. Metzger and M. Heidelberger. Journ. Am. Chem. Soc., *32*, 642 (May, 1910).

No. 177. The Action of Nitric Acid on Triphenyl-methane. By Robert Swarz. Journ. Am. Chem. Soc., *31*, 848 (July, 1909).

DEPARTMENT OF CIVIL ENGINEERING.

(From the annual report of the head of the department.)

The annual report of the department is affected this year by so many changes in the personnel of the departmental force that it may be best to make a somewhat complete statement of the general features of the course in civil engineering as given at present, including the course in sanitary engineering.

On March 15 Mr. C. E. Morrison, instructor in civil engineering, resigned his position to enter the practical work of civil engineering, and at the end of the year Adjunct-Professor Ira H. Woolson closed his connection with the university for the same purpose. The adjunct professorship vacated by Professor Woolson will not be filled, but Mr. J. S. Macgregor, formerly assistant in the materials testing laboratory, has been promoted to an instructorship in civil engineering, and will in the future have immediate charge of the laboratory. The position vacated by Mr. C. E. Morrison has been filled by Mr. J. P. J. Williams, who has accepted an assistant professorship for that purpose.

Mr. L. D. Norsworthy declined re-appointment to the position of assistant to accept an assistant professorship in the Iowa State College. These changes make it necessary to fill two vacant assistantships in the department, candidates for which will be recommended as soon as they can be found.

When the Department of Civil Engineering was established seventeen years ago and when my connection with the University began at that time, the course in civil engineering was radically re-created, so as to constitute a course of educational training for the profession of civil engineering both in its widest historical sense and to meet the advanced and exacting technical requirements of the present time. This necessitated a selection of subjects covering, at least in a general way, the wide field of all engineering not military in character. That plan has scrupulously been adhered to and developed to the present time.

It was then believed that a professional course in engineering instruction could best be based upon, or preceded by, the general educational training afforded by a regular college curriculum. That fundamental

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idea was set forth at considerable length in a paper presented to the International Congress of Education, held in Chicago in 1893, and it was the forerunner of the six year course composed of three in the college and a subsequent three years in the professional engineering school, which was considered and set forth by the faculty of the former School of Mines, prior to the creation of the Faculty of Applied Science as now existing. That admirable plan was then made much of, but unfortunately it has been permitted to lapse into little more than a lifeless though formal recognition of a recommended college preparation in the present announcement of the Schools of Mines, Engineering and Chemistry. I cannot help feeling that at least so far as the Department of Civil Engineering is concerned we lose greatly in true educational breadth and depth by not formulating this course of real professional training so as to make it aggressively prominent in our announcement as the end toward which we are moving as an educational body, and which we are bound to reach as soon as the means and the opportunities of the University may justify the formal prescription of such a course of training, precisely as the Schools of Law and Medicine are now doing.

A period of six years from the time of college entrance to professional graduation is not too long for the present condition of the community as existing in this country. In fact, it is less than the great majority of all complete graduates in law, medicine, or theology require and take. There is no other procedure through which the really educated young professional man may be produced. The broad general training of the college course is an essential requisite for the young man best fitted to prosecute his subsequent profession, and it should not be emasculated by attempting to fuse it into a kind of meaningless composite result by making it a part of his professional course. Each requires that characteristic treatment which will give it its highest efficiency. The college subjects are less technical and more formative in their educational effects and hence are best adapted to the years prior to the professional study. The technical subjects of the latter can be acquired and digested with distinctly greater efficiency after habits of mental acquisition have been developed and established by the prior general training. A prescribed amount of professional educational training can be given more effectively and in less time as a part of a completely developed six years course, than as given at the present time without the general training of the college.

The course in civil engineering, as originally laid out seventeen years ago and as now administered in Columbia University, is intended to fulfill the requirements of such a well-rounded and complete professional training, and neither it nor any similar course can ever reach its full effectiveness until it can be administered under the favoring conditions of a completely developed professional school, nor can any such course of educational training reach its full university significance under any other condition.

The administration of the Department of Civil Engineering is so conducted as to produce at the same time an educated man and a

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member of a broad technical profession. Manifestly the first of these purposes cannot be satisfactorily attained until the general education of the college shall precede the technical training of the professional school. The latter therefore has been developed not beyond its proper proportions but far out of proportion to the former, and that rather one-sided professional development must continue until the opportunity for the prior general education be ultimately attained.

The first year in the course in civil engineering is identical with that of the other courses given in the Engineering School. It is composed almost entirely of general scientific and technical subjects, such as the more elementary subjects of physics, chemistry, mathematics, and draughting, which are not markedly technical and frequently are taken as electives in college courses leading to the bachelor's degree. Wisely a part of this year's training, although a relatively small part, consists of afternoon work in the laboratory and the draughting room, a forerunner of the practical work of the more technical subjects in the three succeeding years. In fact, this first year's work is in essence preliminary or preparatory to the real technical training of the following three years.

The subjects of the second year in the course in civil engineering include in the morning lecture hours, the advanced mathematical subjects of differential and integral calculus, followed by analytic mechanics; the application of draughting to a wide range of work, including shop plans; fundamental ideas in steam power engineering; electrical engineering; excavation and tunneling; introduction to public works in the course on roads, streets and pavements; the metallurgy of iron and steel; and shop work. Probably the most effective part of technical instruction is brought into full force with this year's work, i. e., extended practical application of class-room results to a considerable variety of actual engineering work throughout the afternoon hours in the draughting room, in the various laboratories, and in the shop. While the afternoon hours are not quite so fully occupied in this kind of work during the second year as in the third and fourth years, the second year instruction in the Department of Civil Engineering is intended to, and does, include the test of a student's knowledge by his actual use of it throughout the second half of the working days of this period. It constitutes the introduction of the student to the actual professional work of the engineer.

The third year's work is a substantial development along the lines first fully followed in the second year. Advanced instruction is given in such technical subjects as railroad surveying; the sanitary treatment of water supply and sewerage; the elasticity and resistance of materials in their applications to engineering structures and machines; the theory of trusses, both analytic and graphic; masonry structures, including the theory of earth pressure; the design and construction of retaining walls, arches, tall chimneys, and high masonry dams; the fundamental theory of heat energy and its application in steam, gas and other heat engines; extended electrical engineering work in the development of electrical power and its treatment, distribution and control; besides less extended

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subjects, such as kinematics and machine design. During this year the class room exercises in the morning hours are supplemented by practically all the afternoons fully employed in actual application to a great range of such engineering work as constitutes the principal business of the offices of practicing engineers.

The same general observations are even more pertinently applicable to the subjects of the fourth year, including the entire fields of hydraulic work; foundations; the design, construction, and erection of bridges and buildings; railroad engineering; machine design; sewers, river and harbor improvements; the electrical engineering of railway and power plants, and hydraulic power machinery; besides the thesis work which in every case covers not only analytical or theoretical consideration, but the actual design and construction of work, together with final and complete estimates of cost. The features of the actual doing of professional engineering work in connection with all the subjects covered by class room operations is even more marked in the fourth year than in the third.

The development of effective methods of instruction must in my judgment lead to more extended doing of actual work in the application of analysis and theory to the operations of the practicing engineer. There is no test of a student's knowledge equal to that of its actual use as he will have to employ it in his subsequent professional career. The abstract demonstration of formulæ as an independent operation divorced from their useful applications, which was almost the entire system of instruction not many years ago, is largely a fruitless and fatiguing operation. Experience has shown that the power to use intelligently and effectively any analysis, formulæ, or theory involves their complete understanding by the student, and that the abstract instruction in pure theory in the class room may be reduced to an exceedingly small minimum to advantage, provided it is replaced by the actual technical work of the student in which he must make effective application of such theory or analysis.

In the fourth year, as in the preceding two wholly technical years, the afternoon hours are not only fully and even intensely occupied in the actual operations of design and construction, but these operations are introduced as much as possible in the morning class room work with results which are highly advantageous to the student in every particular.

Although these statements have been made in connection largely with other subjects than surveying and railroad engineering, the latter work as conducted in the summer school at Camp Columbia is entitled if anything to chief emphasis as to practical work. All classes of surveying covered by the course are given there by means of the most complete field operations, I believe, which are to be found in connection with instruction in surveying in this country, or any other country.

The thesis work of this department is intended to be a final exercise or test to both the student's knowledge in the engineering subject which he may elect to treat, and his capacity for initial effort. He is encouraged and in fact required to make this final paper a result of

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his own intelligent initiative. While the instructors of the department aid him along general lines and stand ready to give him such information at all times as they may possess, he must take the lead at all points in digesting and employing such information as he may have obtained either from the instructors of the department or from engineers engaged in similar work to that which he treats, or by technical reading or by any other means. In other words, he is made to feel that the responsibility of the application is wholly his own, precisely as if he were performing the service for a client.

All these statements are as pertinent and applicable to the course in sanitary engineering as to the regular course in civil engineering, although some of the subjects in the former are different from those in the latter.

The entire work of the department, conducted in the manner which has been set forth, is intended obviously to give first of all a thoroughly laid technical foundation for a subsequent professional career, and it is believed that that end is effectively attained. At the same time it is my belief that such a fundamental and general course in civil engineering study, as that conducted in this department, is most admirably adapted to the education of a large number of men who follow callings not of a strictly professional character, but in which they have to deal more or less with engineering matters, such as in the various fields of industrial manufacturing and in the administration of public works, as well as in numberless other similar callings. It has come to be recognized widely that many municipal and other government officials dealing with the administration of such public works as water supplies, a great range of sanitary works, roads, streets and pavements, can best discharge their duties to the public through a thorough knowledge of the engineering features of their responsibilities. In fact it is practically impossible to discharge such duties satisfactorily without that knowledge. Within the past three or four years a commissioner of bridges, a commissioner of docks and ferries, and one of the commissioners of water supply of the City of New York, have been active practicing engineers, and a large list of similar officials could be named for all the more important cities in the country. This field of application of civil engineering education has not yet been as fully recognized as its importance requires, and the efficiency of that education for such purposes will be greatly enhanced when the professional training may be preceded by the general college course.

W. H. B.

DEPARTMENT OF MECHANICAL ENGINEERING.

(From the annual report of the head of the department.)

1. *Amount and Distribution of Instruction.*—This department offered, according to the announcement for the past year, forty-seven courses of one term length, of which thirty-nine were of undergraduate and eight of graduate grade and all of which were given except one of undergraduate and six of graduate grade. Those courses that were not

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given were in every case without applicants. Taking account only of the assigned hours for the courses given, the number of student hours of service rendered by this department for one term was 89,145 as against 90,345 last year. This difference is due to the fact that last year some of the courses were taken by double groups of students on account of change of curriculum common to all the departments of the school. This year the work is on a steady basis and with one exception no course is taken by double groups to accommodate the schedule of attendance. Of the amount of instruction given, 49 per cent. of the student hours was taken by candidates of the degree of M.E. while the other 51 per cent. was given to candidates for other degrees. This indicates that a little less than half of the instruction given by the department was internal, the division between internal and external being about the same as it was last year. With regard to the character of the work: 52 per cent. of the student-hour service was by lecture and recitation, that is to say, in class room, while 48 per cent. was by the laboratory or drafting room method. Of the lecture and recitation work alone 39 per cent. of the instruction was internal and 61 per cent. external, while the laboratory and drafting-room work was divided inversely, 60 per cent. being internal and 40 per cent. external. It will appear from these figures that the department is rendering a very considerable service to all the other departments and its requirements of space, equipment and instructing staff are not to be judged by the number of students who are candidates for the Mechanical Engineering degree.

STUDENT HOURS FOR ONE WEEK.

Mechanical engineers:	Lect.-recit..... 1,243;	Lab.-draft.....	1,710
Other students:	Lect.-recit..... 1,892;	Lab.-draft.....	1,098
Total student hours for one week			5,943
Total assigned student-hours for one term given to candidates for M.E. degree, lecture and recitation			18,645
Total assigned student-hours for one term given to candidates for M.E. degree, laboratory and drafting room			25,650
Total assigned student-hours for one term given to candidates for other degrees, lecture and recitation			28,380
Total assigned student-hours for one term given to candidates for other degrees, laboratory and drafting room			16,470
			89,145

No credit has been allowed in these figures for conferences with students, visiting engineers and educators, prospective students nor for reading, correcting and marking thousands of laboratory reports and student note books, for the writing of lectures, conducting of correspondence with manufacturers, collection and filing of data, writing of papers of a professional character, university committee work, service in professional societies, reading of transactions and professional journals, inspection and testing of machinery or for research. All these things require time and all these things must be done. To carry out the work that has been done has required a personal sacrifice on the

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part of every member of this department, each of whom has served every day and longer hours than any man should be asked to work. In fact so severe has been the tax upon the staff that unless it is relieved the men will become inefficient as instructors in a few years as they must inevitably fall behind in their knowledge of current professional practice.

2. *Practicing Engineers as Lecturers.*—The plan of using practicing engineers as lecturers for the presentation of subjects in a manner better than could be accomplished by the ordinary staff of instructors and professors has been continued and has been so unqualifiedly successful that the plan can never be replaced by any substitute. These men have served in all cases without pay and while their value to the university cannot be measured in dollars yet it is unfair to expect them to continue to so serve nor can we count on their enthusiasm to sustain their interest. Means must be found without delay for properly and adequately paying these men for the services they render not only to the university but to the cause of high class professional education, through the university. In this connection it is only fair to note that Harvard, using some of the same men, is reported as paying fifty dollars per lecture and expenses. Last year the manufacturers of elevating and conveying machinery supported one one-hour course of instruction on that subject. This year they withdrew their support, believing that their duty was done in helping to establish the course and that the university should now take up the burden. In the following will be found the names of the lecturers who have served and the course in which their lectures contributed.

M.E. 53—*Steam Turbines.*—Mr. C. E. Stone = 1 hour; Mr. C. L. Loewenstein = 4 hours; Mr. H. S. Reist = 1 hour; Mr. F. T. Erben = 1 hour; Mr. R. H. Rice = 1 hour; Mr. C. B. Edwards = 1 hour; Mr. J. G. Callan = 4 hours.

M.E. 52—*Principles of Machine Manufacture.*—Mr. F. W. Miller = 1 hour; Mr. A. Spangenberg = 2 hours; Mr. C. Gingrich = 1 hour; Mr. A. Aikmann = 1 hour.

M.E. 52—*Works Management.*—Mr. C. B. Going = 6 hours; Mr. C. U. Carpenter = 4 hours; Mr. H. L. Gannt = 2 hours; Mr. W. McFarland = 1 hour; Mr. H. Emerson = 4 hours; Mr. R. Lingley = 3 hours; Mr. E. J. Prindle = 1 hour.

M.E. 62—*Refrigeration Machinery.*—Mr. Fred Ophuls = 14 hours.

M.E. 64—*Air Machinery.*—Mr. F. W. O'Neil = 16 hours; Mr. H. DeB. Parsons = 2 hours; Mr. J. G. Callan = 2 lectures.

M.E. 66—*Elevating and Conveying Machinery.*—Mr. L. DeG. Moss = 13 hours.

M.E. 68—*Manufacturing Plant Design.*—Mr. Chas. Day = 6 hours; Mr. L. DeG. Moss = 2 hours; Mr. G. K. Hooper = 2 hours; Mr. P. Moses = 2 hours; Mr. E. R. Hardy = 2 hours; Mr. W. Ballinger = 2 hours; Mr. H. F. Stimpson = 4 hours.

M.E. 70—*Pumping Machinery.*—Mr. C. A. Hague = 13 hours; Mr. L. C. Loewenstein = 3 hours.

3. *Efficiency of Instruction.*—Attention was called last year in my annual report to the necessarily low standard of efficiency of instruc-

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tion that results from too few recitations, too many lectures, recitation classes too great in numbers, too few laboratory and drafting room instructors for the number of students at work, and in addition from the fact that all instructors are worked so hard as to be continually under the stress of fatigue. In view of the conditions that have prevailed it is surprising that we have done as well as we have but it is unfair to expect good results to continue and it is doubly unfair when we claim either by expression or inference to give the best instruction to be had. There is most emphatic need for more instructors in this department but as there seems to be a similar need in other departments in which students who are candidates for the degree of mechanical engineer are required to study, it has been deemed best to take up the subject on broad lines rather than to merely urge the appointment of more men in one place. Accordingly, a general plan for limiting the assignment of professors with regard to number of hours service and number of students to be taught in each hour of assignment has been drawn up and presented to the Committee on Instruction for its criticism, analysis and study with the hope that from this suggestion the committee will be able to report to the faculty a comprehensive scheme of limitation of assignment to be applied to the whole school in the interest of efficient instruction. While it is true that the carrying out of any such plan will in all probability increase the staff in the various engineering departments it is believed that if a similar plan be applied to the whole university the total expenditures on salaries need not necessarily be much increased. This, however, is only a supposition and may not be true.

While in the interest of efficient instruction the most pressing need lies in the appointment of more instructors yet this alone will not accomplish the results because without suitable equipment and sufficient room space these instructors cannot efficiently meet their students or do their own work. The equipment and space now occupied by this department is woefully inadequate. It is time that it was recognized that in shop facilities we have practically nothing and that the mechanical laboratory that has been so often characterized as a magnificent equipment is nothing of the sort. It is very far, indeed, from containing the apparatus necessary for demonstrating the important principles underlying the operation of standard machinery, which can be taught only partly in class and can never be really understood without experimental demonstration. The present laboratory space of 13,445 square feet is divided as follows: West Vault 10,400 sq. ft., instrument laboratory, room 207, 1,645 sq. ft., gas and oil engine laboratory, 800 feet, shop and store room, 500 feet. For the six men constituting the staff of the department there is only one office of about 225 sq. ft. The drafting room, accommodating the second, third and fourth year students of this department as well as students of other departments, as shown by the list of courses, has a total of 2,365 sq. ft., in rooms 506 (1,040) and 510 (1,325). Means for filing the students' work between periods, storing his instruments and accommodating his drawing board are to say the least inadequate and so poor is the protection offered as

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to discourage the men who are trying to do fine work. There is practically no means for filing reference drawings of which thousands are needed for comparative study by men engaged in designing. For storing appliances and characteristic pieces of apparatus to be used in the demonstration of the lectures there is practically no space as all the facilities are confined to a single glass case on the third floor, which, together with a little surrounding floor space, does not exceed 180 sq. ft. To adequately present to the students engaged in studying the various branches of machinery we should have not less than 100,000 catalogues for student reference. Such as we have at the present time are confined to one small room of about 135 sq. ft., not more than one tenth of the space actually needed and entirely without suitable filing means and personal attention to make that little useful. Taking all the space together, that used for laboratory, offices, drafting room, museum, catalogue library, the department is confined to 16,350 sq. ft. The department is absolutely without shops suited to the needs of the courses of student because the little manual training shops of Teachers College are not only inadequate but harmful in as much as they are not representative of the standard equipment necessary for the manufacture of machinery. There is no departmental library for students to consult references and no place for them to sit down and study or work out problems, much of which work, if not all, should be done here and not at home.

Your attention was called to these matters by my letter of December 1, 1909, pointing out these needs in detail and requesting a new building for which a tentative plan was submitted, together with a list of rooms, space and equipment suitable for the conduct of the work of the Mechanical Engineering Department of the largest university in America. It is hardly necessary for me to repeat what was then said except to point out the conclusions. According to the plan submitted for providing space for pattern shops, machine shop, forge shop and foundry there would be required 34,200 sq. ft., and for the extension of the laboratory about 17,000 sq. ft. in addition. For the exhibition, suitable for student study, and storage of mechanical apliances and standard machinery necessary for lecture demonstration, about 10,000 sq. ft. is required. For drafting rooms and the filing of drawings and catalogues, about 10,000 sq. ft.; for a departmental library to contain the literature on the subject and a study room where students may prepare their lessons, solve their problems and consult their instructors freely, about 6,000 sq. ft.; for offices about 5,000 sq. ft., and all of which totals up to a space requirement of 82,200 sq. ft. as compared with the present occupied square feet of 16,350.

It is sincerely hoped that means will be found to supply the space and equipment on the one hand, and a proper number of instructors on the other to render our instruction efficient and a credit to the largest university in the largest and richest city in America.

C. E. L.

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EDITORIAL.

With the first number of Vol. 32, the task of editing and managing the QUARTERLY devolved upon the present editors and it would seem proper at this time to outline what they consider to be the policy of this publication and their duties in connection with it.

First of all the QUARTERLY is the official organ of the Alumni of the Schools of Mines, Engineering and Chemistry, and as such ought to contain primarily material from that body.

Secondly, the QUARTERLY ought to lay before the graduates the current work of the schools, especially the research work performed in the various departments and keep them in close touch with Columbia.

This can be done very effectively if every alumnus will make it a point of duty to contribute something, no matter how small, of interest to our readers. We ought to have such an abundance of contributions that instead of having to plead and beg for articles, there will always be material ahead so that the issues of the QUARTERLY may be brought out on time. Many will answer that they have nothing of sufficient importance to make an article. Granting however that in some cases this may be true, everyone has something of interest enough to go into a column of "Notes from the Alumni" which it is intended to start as soon as the notes materialize.

In regard to the current work on the Campus, whilst this is partly taken care of now in the various departmental reports, the accounts ought to be more complete. This can be done through the aid of the various departments if they will send us short abstracts of new work that has been completed (even though it is published fully elsewhere), of lectures that have been given, of changes in the courses and so forth.

In conclusion we must emphasize the fact that we are merely the Editors—we edit what is sent in to the best of our ability and the praise or blame of the QUARTERLY must be borne by the Alumni. If everyone lends a hand there is no reason why the QUARTERLY cannot fulfil its purpose and be the best of its kind anywhere.

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BOOK REVIEWS.

Practical Shaft Sinking. By FRANCIS DONALDSON, M.E., Chief Engineer, The Dravo Contracting Co. 12mo, 143 pages, 63 figures. New York, McGraw-Hill Book Co. $2.00.

The author of this book possesses a rare combination of abilities: he is master of his profession, a business man, and yet writes with fluency, clearness and accuracy. His obvious purpose in writing, furthermore, is to disseminate first-hand information rather than to bury a few grains of it amid pages of ill-digested and inconsequential details borrowed from other sources. And yet, the details, to overlook which brings ruin to any engineering project, are by no means lacking; they are merely so concisely presented that every word counts. Some of the little points, by the way, we do not remember to have seen in other books on the same subject.

The following topics are discussed: I., Basis for shaft-sinking contracts; II., Plant required for shaft sinking; III., Sinking through surface soil, with particular reference to sheet-piling, forepoling, caissons, and drop-shafts; IV., Pneumatic caisson method for soft, watery soil; V., Sinking in hard rock, with details about drilling and charging the holes; VI., Special German methods for soft, watery material, such as sack-boring and the freezing process; VII., Kind-Chaudron and cementation processes for fissured rock; VIII., Lifting water from shafts in process of sinking; IX., Shaft linings of wood, masonry or metal; X., Concrete shaft linings.

One highly commendable feature of the book is that the author does not hesitate to express his own opinions, as, for example, in advocating pumps of the horizontal instead of the vertical type for sinking work. When we reflect on the years of experience that have led up to these opinions, we can only wish that more engineers of Mr. Donaldson's caliber would take the trouble to publish their notes, data and suggestions, and to put them into readable form.

E. K. J.

Compressed Air Plant. The Production, Transmission and Use of Compressed Air, with Special Reference to Mine Service. By ROBERT PEELE. Second Edition, Revised and Enlarged. New York, John Wiley & Sons: London, Chapman & Hall, Ltd., 1910. 8vo, pp. 502, 209 figures. Cloth, $3.50 net (15/- net).

When the first edition of Professor Peele's work entiled "Compressed Air Plant for Mines" appeared two years ago, the literature on this subject was largely confined to sundry papers scattered through the technical journals, and a book covering the entire ground was a much needed addition to an engineering library. So well was this book received that a second edition has now been published, carefully revised and enlarged to the extent of some 135 pages of text and 87

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illustrations, the principal additions being in about 90 pages of text and 63 illustrations relating to the construction and operation of rock drills, coal cutting machines and channelling machines.

The work is divided into two parts: Part First, Production of Compressed Air; Part Second, Transmission and Use of Compressed Air.

In Part First, Chapter I.—Introduction, a brief historical sketch of the use of compressed air and rock drills is given, and then follows a concise comparison of the value of compressed-air transmission of power to that of steam, calling attention to the great work underground that has been made possible by the use of compressed air in connection with high explosives. The comparative merits of compressed air and electricity are also mentioned, and attention is called to the fact that up to the present time the use of electricity for power underground has been hampered by lack of a satisfactory electric rock drill.

Chapter II., Structure and Operation of Compressors, gives in full the various types, including steam, electric and water power compressors, and the descriptions are illustrated by numerous cuts and working drawings. Chapter III.: The Compression of Air—Outline of Theory. Chapter IV.: Wet Compressors. Chapter V.: Dry Compressors. Chapter VI.: Compound or Stage Compressors—Theory and Operation. Chapter VII.: Air Inlet Valves. Chapter VIII.: Discharge or Delivery Valves. Chapter IX.: Mechanically Controlled Air Valves and Valve Motions. Chapter X.: Performance of Air Compressors. Calculation of Horse-Power of Single-Cylinder and Stage Compressors. Chapter XI.: Air Receivers. Chapter XII.: Speed and Pressure Regulators for compressors—Air Cylinder Unloaders. Chapter XIII.: Air Compression at Altitudes above Sea-level, Relation between Compressor Output and Barometric Pressure. Chapter XIV.: Explosions in Compressors and Receivers. Chapter XV.: Air Compression by the Direct Action of Falling Water.

In this part the descriptions of the performance of air compressors in Chapter X, giving the method of reading and calculating the power of the various types, are excellent and valuable field tests with diagrams and tables of power and costs are given on pages 176–89 inclusive, which will be found most useful.

Part Second, Chapter XVI.: Conveyance of Compressed Air in Pipes. The losses of power due to various causes are discussed and formulæ for determining these losses with useful tables of reference are given. Chapter XVII.: Compressed-Air Engines—Working at Full Pressure or with Partial or Complete Expansion. Chapter XVIII.: Freezing of Moisture Deposited from Compressed Air. Chapter XIX.: Reheating Compressed Air. Chapter XX.: Compressed-Air Rock Drills. Chapter XXI.: Compressed-Air Hammer Drills—General Construction. Chapter XXII.: Coal Cutting Machines. Chapter XXIII.: Channelling Machines—Applications and General Construction. Chapter XXIV.: Operation of Mine Pumps by Compressed Air. Chapter XXV.: Pumping by the Direct Action of Compressed Air—Pneumatic-Displacement Pumps. Pohle Air-Lift Pump—Theory and Operation.

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Chapter XXVI.: Compressed-Air Haulage for Mines—Construction and Operation of Compressed-Air Locomotives.

In the second part, Chapters XXI., XXII., XXIII. are entirely new. The records of work done in actual practice given in Chapter XX. for rock-drills, and in Chapter XXI. for hammer drills are especially interesting and this is data which has not before been published.

In reading this book, one is struck by the comprehensive manner in which subjects are presented and the clearness and completeness of the descriptions, while at the same time, unnecessary and superfluous detail is conspicuous by its absence. The book is exceptionally well printed and the illustrations are numerous and clear. A well-arranged index of 13 pages at the end adds greatly to the ease with which reference can be made to any desired subject.

A. L. W.

Botany and Pharmacognosy. By HENRY KRAEMER. Philadelphia, Pa., J. B. Lippincott Co. September, 1910. Pp. viii + 888, f. 1—344.

The present fourth revision of Professor Kraemer's book maintains the same order of treatment as the previous edition. Material changes, however, have been made throughout the work, many topics being rewritten and much additional matter has been incorporated both by way of illustrations and text. Twenty-three new figures and forty-eight pages of text have been added.

The first part of the book deals with the classification and morphology of plants. A chapter upon the cultivation of medicinal plants is also introduced. Necessarily the consideration of so extended a field within the limits of 400 pages must lead to a brief presentation of the various topics. This doubtless compelled the author to disregard the relationships of the groups and the alliances and to frequently overlook the significance of structures and their functions. While we recognize that the mechanical side of the plant is the important feature to the pharmacist, we can not but regret that more attention has not been given to the lifeside of the plant, even at the expense of some of the formal presentation.

The second part, dealing with pharmacognosy, treats of crude and powdered drugs and their methods of analysis. The presentation of this subject is a valuable contribution to the knowledge of pharmacognosy. The author's study and classification of powdered drugs is so well known as to require no further comment, but in the present revision he has supplemented this work by devoting forty pages to the micro-analyses of the active and other constituents of drugs that may be obtained in crystalline form. Heretofore the pharmacist had to rely largely upon a study of the tissues in determining the character and purity of drugs. The petrographical methods of study of some of the important plant constituents presented in Part IV. will be of great value in the detecting and in the accurate study of these substances and it will also serve as an important supplement and check upon the ordinary microscopic and chemical examination.

This work reflects some fifteen years of effort on the author's part,

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both as student and teacher and we feel confident that it will meet with wide approval both as a text for the class-room and as a practical guide to the pharmacist in the microscopical examination of vegetable drugs and allied products.

CARLTON C. CURTIS.

American Red Cross Abridged Textbook on First Aid (Industrial Edition). By MAJOR CHARLES LYNCH and FIRST LIEUTENANT M. J. SHIELDS. Philadelphia, P. Blakiston's Sons & Co., 1910. Pp. 175, 49 ill., 4⅛ × 5⅞. Price 30c.

This compact little volume should be in the hands of every engineer. The value of knowledge of first aid in emergencies is shown by the following statement in the introduction: "In certain mines in Pennsylvania first aid to the injured has been well and generally taught to miners. This instruction and this alone has resulted in reducing sick and death benefits 50 per cent. or one half."

The book contains ten chapters dealing with the Structure and Mechanics of the Body, First Aid Materials, Wounds, Sprains, Dislocations, Fractures, Burns, Injuries from Heat, Cold and Electricity, Unconsciousness, Poisoning, How to Carry the Injured, Industrial Accidents, and Organization of First Aid Instruction.

The treatment is simple and clear, much of the technical and superfluous material included in books of this kind has been eliminated, and the text is properly illustrated with cuts and diagrams.

The only defects worth mentioning are the absence of any reference to drowning accidents and the undue emphasis given to the Sylvester method of artificial respiration, which is far less efficient than the new prone-pressure method.

GEORGE L. MEYLAN.

Rock Drills, Designs, Construction and Use. By EUSTACE M. WESTON. $4. McGraw-Hill Book Company, New York.

A more appropriate title for this work would have been "Rock Drilling by Machine" since nearly three-quarters of the book, and much the most valuable part of it, deals with the subject of drilling, the construction of the various drills being reviewed in the first 100 pages.

Chapters II. and III., treating respectively with the mechanical details of standard piston drills and air-hammer drills, are similar, but not equal, to the chapters on corresponding subjects in Professor Robert Peele's latest book on compressed air. Chapter IV., on electric drills, is a good resumé of the latest advances in that difficult problem. Much of the information in these first three chapters, as the author very frankly admits, is compiled from manufacturers' catalogues.

Beginning with Chapter V., however, the author is on more personal ground. Having gained his information at first hand by long experience in the manipulation and supervision of machine drills, his conclusions as to the most economical manner of using them, the most satisfactory kinds of bits, and the reason for the shape of their cutting

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edges, are extremely valuable. Likewise, in Chapters X. and XI., dealing with explosives and blasting, the author's precepts, based on his own experience, are of much greater practical value than the erudite quotations from other works. In Chapter XII., containing examples of rock-drill work, the author records his own observations in South Africa and Australia, but for examples elsewhere in the world, he is naturally driven to accept the statements of other students of the subject, published here and there in the literature of mining.

The compiling of data on a given subject for condensation into one volume, is always a thankless and undervalued task, but in our opinion, our education in rock drilling would have been better served if Mr. Weston fortified with years of actual, hard earned experience, which the professional book writers seldom possess, had restricted somewhat the scope of his work and confined himself to obervations and conclusions of his own. This authoritative information is just the kind that is so hard to find, since those who are best qualified to teach frequently lack either opportunity or inclination to exercise their ability to write.

E. K. J.

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THE SCHOOL OF MINES QUARTERLY.

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THE

SCHOOL OF MINES QUARTERLY.

VOL. XXXII. APRIL, 1911. No. 3.

NOTES ON LABOR AND MINING COSTS IN THE SOUTH AFRICAN MINES.

BY ROBERT PEELE.*

The labor question in the South African mining districts has been a rather difficult one, especially in recent years. Up to the time of the Boer war, which began in the latter part of 1899, nearly all of the underground work, and most of that on the surface, was done under white supervision by Kafirs, comprising Basutos, Zulus and Shangans, together with some natives from Mozambique, and other regions to the north and on the east coast.

On reopening the Transvaal mines after the war (about May, 1901), the scarcity of laborers seriously retarded the full resumption of operations, and it was not long before it was proposed (first in Rhodesia, afterwards in the Transvaal) to introduce Asiatic labor. Strong opposition to this innovation manifested itself, but it was eventually tried and large numbers of Chinese were imported under contract. After several years, however, during which the Chinese proved to be fairly satisfactory workers, the opposition to the continuance of the plan became so strong, both in Great Britain and parts of South Africa, that all the Chinese were returned to their own country, as fast as their contracts expired.

At the present time, laborers for fixed terms of contract are obtained by the mining companies chiefly through the aid of the native Labor Department—of the Johannesburg Chamber of Mines—which was established about 1902. Agencies have been

* Professor of Mining, Columbia University.

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opened among the different native tribes for the recruiting of "boys," mainly for underground work, and there is now comparatively little difficulty in securing a sufficient supply, though at a greater cost to the mining companies, who pay to the labor department a bonus for each "boy." Some of the companies have their own private and unofficial agents; which, in a measure, would appear to be a return to the old, objectionable practice of indiscriminate "touting" for recruits among the tribes, a practice which resulted in greatly increased cost to the mines. As the natives will rarely work except under some form of compulsion, contracts are made with the tribal chiefs, to supply them for three- or six-month terms.

White labor in South Africa has also constituted a vexed question. In all parts of the world where black labor is employed, the whites are less efficient, as "laborers," than where this condition does not exist. On the Rand, and at Kimberley and elsewhere, the "hard work" underground, such as hand-drilling and tramming, is invariably done by the natives. Immense areas of the Rand reefs, within the limits of the present workings, are quite flat—say between 18 and 35 degrees to the horizontal. And, as the reefs themselves are often much too thin to permit the use of machine drills, hand-drilling is essential. In the thinnest parts of the South Reef and the Main Reef Leader it is often necessary to keep down the stoping width to the minimum (18 to 22 inches) in which a hammer can be wielded. While drilling in such stopes the "boys" have to lie on their sides, or sometimes actually on their backs. This work is hard and exhausting and white men would never consent to do it, where native labor is at hand. With a few months of practice in work of this kind, many of the Kafirs become surprisingly expert and efficient, when under the supervision of good shift-bosses. Even in running machine drills, the "boys" generally shift, set up and operate the machines, the holes being pointed by white men. Frequently, for stoping, driving, etc., either by hand or machine drilling, contracts are taken by white miners, who direct the native workers, they themselves being under the supervision of mine foremen and shift-bosses. In this way, a single white man may have charge of several machines, or a large gang of "hammer boys." When drilling by hand, a good "boy" will

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readily put in 33 to 36 inches of hole per shift, in banket reef of average hardness; with stoping machines, say 30 to 35 feet per shift.

Before the war this contract system was carried on much more widely than now, and energetic white miners were sometimes allowed extensive and very remunerative contract privileges. It is said that such men frequently made as much as $400 or $500 per month. At the present time these opportunities do not exist. The wages of the natives have not undergone much change, but those of the white miners have been reduced. This result has been brought about, amongst other things, by the necessity of cutting down mining costs, due to the general impoverishment of the Rand reefs in depth. Only a few years ago the mining cost per ton—not including milling nor general expenses—often ranged from $3.00 to $3.25; now, some of the deep mines are working at $1.85 to $2.15 per ton of ore broken. This large reduction has been made possible only by efficient underground management and carefully looking after small economies. I may add, for purposes of comparison, that in 1896, at the Simmer and Jack Proprietary mines, on a production for the year of 156,930 tons, the total average cost for mining, milling and general expenses, was 28.337 shillings (=$6.80) per ton. In 1898, at the same mine, on a production of 183,240 tons for six months, the total cost was reduced to 19.883 shillings (=$4.75). During the same period the cost at many less well managed properties was considerably higher.

At the Dutoitspan mine, Kimberley, which may be taken as a representative mine of the De Beers Diamond Mining Company, and at which about 240,000 mine car loads (=192,000 tons) are hoisted per month, the total mining cost of the "blue ground," delivered at the surface and ready for treatment, now ranges from 2 shillings 5 pence to 3 shillings (58 to 72 cents) per ton. Of this amount, the underground cost alone—including breaking, timbering and transport to shaft, is about 1 shilling 5 pence (34 cents). Nearly all the underground work is done by natives in charge of white men. The ground is generally broken by Ingersoll-Rand 2¼-inch drills, averaging 34 to 40 feet of hole per 8-hour shift; and Waugh hammer drills, averaging 60 to 70 feet of hole per shift. In all the five mines of the De Beers Company, mechanical trans-

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portation is employed underground, either by electric locomotives or the overhead endless wire rope system. Side-dump cars, holding 16 cu. ft. (1,600 lbs.), are tipped into the shaft-pockets, either automatically or by hand, and hoisted in vertical shafts by 10-ton skips. To handle the immense tonnages at these mines, considerable labor is required, both at bottom and top of shafts, even with the automatic appliances; and for work of this kind, which becomes almost rhythmic in its continuity and regularity, the native boys are singularly well adapted. The mine cars are conveyed from the mouth of the shaft to the "weathering floors" by over-head rope haulage, costing at the Dutoitspan mine 3.5 cents per car load, or 2.8 cents per ton, for an average haulage distance of four miles.

The "compound" system of South Africa is an interesting one. Owing to the semi-civilized condition of the natives, and the fact that nearly all of those working in the Transvaal and at Kimberley live in other parts of the country, they are housed in "compounds." These are extensive, continuous ranges of small, one-story buildings, usually erected around large, open rectangular areas. The outer walls are often blank—sometimes provided with small windows—and the doors open on the interior yard. Thus, the "houses" are merely compartments. in each of which there are double or triple decks of rough bunks around three sides. Each compartment or room ordinarily accommodates from twenty to forty boys, and several thousand are sometimes housed in a single compound. At Kimberley, at least one of the immense compounds houses from 2,500 to 3,500 boys, the interior yard comprising an area of many acres.

On the Rand, rations of food are supplied to the boys, either cooked or uncooked, as they prefer. At meal-times they form in lines and receive their cooked rations in their tin pans and dishes; or cook their food in their own way, in the individual rooms or before the doors, with the aid of improvised fire-places. On the whole, they live much better than they are accustomed to at home.

At the De Beers property, each compound of the five large mines is complete in itself, including a store for the sale of groceries, vegetables, clothing, etc., a butcher shop and a hospital (with operating room, infirmary, dispensary, white doctor and surgeon and a staff of nurses—most of the latter being natives). The "boys"

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are furnished with free fire-wood, but not rations. They buy such food and other supplies as they may desire, their wages being adjusted in conformity with this arrangement. The good quality of the articles kept on sale indicates that the standard of living of the natives is being raised by contact with the white man, and that they are learning to appreciate some of the advantages of civilization. At this well-managed property, the hospital facilities are excellent. The operating room, for cases of injury, is perfectly clean and well-kept, and all the necessary hospital supplies are liberally provided. Each compound is cared for by a staff of white men under a "compound manager," who is accountable to the general manager. A total of about 2,000 whites and 16,500 natives are employed at the De Beers mines, when in full operation.

The question of loss of diamonds, through theft, has long been a serious one at Kimberley. The words, "Illicit Diamond Buyer" (I.D.B.) form a common expression in the city and surrounding district, and the local laws against the practice are rigidly framed and enforced. For some time past, a system has been employed by the De Beers Company, whereby any native who finds and delivers up a diamond which he has discovered in the course of his work, receives, as a bounty or bonus, 5 shillings ($1.25) per carat, together with a percentage, on a sliding scale, of the current value of the diamond in its rough state. This amount is credited to his account and settled in cash at the expiration of his term of contract. On the occasion of a visit to one of the compounds, I was shown a diamond of something over 70 carats, worth (uncut) about $650, that morning brought into the office, of which the "boy's" share was about $155, or nearly 25 per cent. of the value of the stone. To a Kafir this is a large sum, and suggested that, at the end of his contract, the lucky finder would certainly return home and stay there. But I was assured that the company would nevertheless benefit by the transaction, because the Kafir would inform all his friends and neighbors, and new recruits would be forthcoming without searching for them. This plan results in saving a large part of the value of diamonds that might otherwise be stolen, and in addition assists the solution of the labor problem by making the natives contented.

However, the materialistic principle that "honesty is the best

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policy," is not yet sufficiently realized by the natives, and at the diamond mines they are invariably confined to the compounds during their contract periods. The compounds are connected with the mine shafts by short underground galleries, through which the "boys" pass to and from their work. These passages terminate at the shaft in a station just below the surface. Furthermore, wire nettings are often stretched horizontally, inward from the eaves of the houses, around the perimeter of the compound enclosures, and of sufficient width to reduce the possibility of diamonds being thrown outside to the care of confederates. On the whole, the "boys" are well cared for and do not object to the compound system, which long experience has shown to be the only feasible mode of housing the large numbers of native workers.

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SOME NOTES ON MINING IN CUBA.

By BENJAMIN B. LAWRENCE.*

Iron mining in Cuba has taken great strides during the past year or two, owing to the discovery on the north coast of chrome iron ores. The ore is found on the surface and extends to an average depth of some thirty feet. It is said to be the result of the alteration and decomposition of serpentine, making a marketable iron ore. The ore may be found a few miles from the coast and covers a very large area. A complete analysis of a general sample gave the following results:

	Per Cent.
Iron (after drying at 212° F.)	46.03
Silica	5.50
Alumina	10.33
Chromium	1.73
Phosphorus	.015
Hydroscopic water	31.62
Nickel	1.04

The hydroscopic water makes the problem of handling this material extremely difficult, inasmuch as the water, being such a large percentage of the total weight, must be first driven off by heat before shipment to the United States. Elaborate plans have been made by the companies operating this deposit to get rid of this water, and afterward to nodulize the product.

A number of interesting articles upon the development of iron ore at Mayari may be found in the technical journals. One company has built a plant capable of handling 5,000 tons of ore per day. Cost of production is given at $1.00 per ton. Mining by steam shovel 11 cents a ton, hauling 15 cents and the balance, 74 cents, is for driving off the water, nodulizing and loading in vessels.

Apropos of iron mining in Cuba, the accompanying photograph (Fig. 1) shows a novel way of loading a vessel by means of a belt conveyor traveling over a long lever-like steel construction, which extends far enough over the water to enable the vessel to be loaded without danger of its being wrecked on the rocky coast.

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Some of the most modern methods of mining can be found at Daiquiri, where there is an extensive deposit of iron, from which the ore has been shipped for more than twenty years. This mine is now producing some 540,000 tons a year of hematite containing 58 per cent. iron, .02 phosphorus and .25 sulphur—a very satisfactory iron ore, and much sought for by blast furnace men.

The accompanying photographs (Figs. 2, 3 and 4) show these mines, and a method of working by benches with steam shovels.

Another extensive mine may be seen near the city of Santiago at Juragua. Here also operations are conducted by an American

FIG. 1. New Method of loading vessel. No harbour.

company along economical lines. The ore produced is of the same general character as other iron ores in Cuba.

Extensive pyrite deposits have also been found on the island. It is interesting to note that the oxidation of these pyrites extends to a depth of from 200 to 300 feet, whereas in a climate where there is less rain—or in colder climates—the depth of oxidation of pyrites is seldom greater than 60 feet. These pyrite deposits are—in the cases examined—replacements of the schists along contact with limestone.

None of these deposits has as yet been opened up commercially, but there is great promise that Cuba will be a heavy shipper of this mineral some day.

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Most of the pyrite deposits found so far have been in the interior of the island, which is for the most part inaccessible, except by trail or wagon roads which are "unfit for publication." In this respect Cuba is far behind the times. Its lack of transportation facilities is a sad illustration of the need of progressiveness on the part of its

FIG. 2. Spanish American Mining Co., Daiquiri, Santiago de Cuba.

inhabitants. The people of the island will stick to the mule as a means of transit, and they are content to devote two or three days to a journey which we would think irksome if it consumed more than a few hours. Owing to the state of the roads, the automobile is barred from the civilizing influence which it has had in other countries. In justice it may be said, that road building in Cuba is an expensive undertaking owing to the heavy rains, which make it necessary to protect a road against being washed away and destroyed during the rainy season. That good roads can be built, however, is evidenced by the one from Havana to Matanzas. In fact, there are a number of excellent roads out of Havana, and one built by General Wood out of Santiago. These roads should be object lessons to the people.

Mining in Cuba is by no means of recent development, as copper ore was produced from the El Cobre Mine at Santiago by Spanish adventurers in 1544. There is every evidence that the first copper

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FIG. 3. Pit at Daiquiri.

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to make its appearance in the western hemisphere came from this old mine. It was owned and operated by the Spaniards during the sixteenth and seventeenth centuries. In Oliver Cromwell's time it attracted considerable attention, and a commission was appointed by this cavalier to see whether it was worth while to quarrel with Spain to obtain possession of the mine, and, ultimately, the island of Cuba. I presume that Cromwell's career ended sooner than he anticipated, otherwise I am afraid that Spain might have had this gentleman to reckon with.

The English later on did come into possession of this property, and through their enterprise, during fifty years, from 1830 to 1880, an enormous quantity of rich copper ore was mined from the property, which was shipped to Swansea in Wales.

Even in those days this property was operated on a very large scale, as is shown by the interesting ruins and by some machinery and appliances which are left. An old Cornish pump which was used on the mine was especially interesting. One wooden rod belonging to this pump was 63 feet long and was made of a piece of timber 18 in. × 20 in. The pump bob weighed 60 tons and was brought over from Glasgow in 1860, and it is said that it occupied the energies of 500 men for six months to transport it from the coast, ten miles up the mountain side to the mine.

Tall chimneys in the vicinity show that smelting operations were conducted at the property on an extensive scale.

The ores produced were for the most part the richer oxidized copper ores containing from 13 to 30 per cent. of copper. As it required a 13 per cent. ore to yield enough to pay for shipment to England, some extremely rich ore must have been shipped to Swansea in the early history of this property.

In 1844 a railway nine miles in length was built from the Bay of Santiago to the mine, which replaced some 2,000 mules which were formerly used in transporting the ore from mine to bay. The Queen of Spain was the principal stockholder, and an arbitrary rate of $6.00 per ton for down, and $4.50 for up freight was established. The railroad ultimately became the controlling feature of the situation, and, it is said, absorbed most of the revenues from the property in freight rates.

It is surprising what deep mining the antiguas did. In the Cobre

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FIG. 4. Mining at Daiquiri.

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Mine they reached a depth of 1,200 feet. The flows of water, especially in the rainy season are enormous, at least 2,000 gallons a minute may be expected to be pumped from El Cobre for two or three months during this season. No one who has not experienced the tropical rains can appreciate the volumes of water that drain into the workings of this mine.

An interesting discussion of the geology of this district has been made by the manager of the property, Mr. E. H. Emerson.

Immediately after the Spanish-American War, American capital took up the problem of unwatering this old mine, and they have as yet failed to reach the lower workings in which ore of good value is said to occur. Large sums of money have been spent in the attempt to unwater the property, but so many difficulties have arisen that it is as yet an unsolved problem. A recent experiment by Mr. Emerson in the use of reinforced wooden pipe, using a jet of compressed air, has been successful. The use of iron pipe is impossible owing to the acid water.

Progress in unwatering and rehabilitating a mine which has been worked to a depth of 1,200 feet, and some 6,000 feet laterally, must necessarily be slow; and meantime, in the upper workings, a great deal of copper ore (too low grade to pay in the old days) has been shipped to the United States with very satisfactory results. The product of the mine is now some 6,000 tons of 5 per cent. copper ore monthly.

There are a number of smaller copper operations in Cuba, but with the exception of the Cobre there are no shipping mines.

The inaccessibility of the country has had a great deal to do with the slow development of Cuba's mineral resources. The island is rich in minerals beyond question, but until its government recognizes the importance of building roads—both rail and wagon roads—many of the richer districts will remain undeveloped.

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THE NEWLY DISCOVERED EMERALD MINES OF "SOMONDOCO."

By E. B. LATHAM.*

The continued increase in the values of emeralds during the past ten years—until at present they outrank diamonds—lends considerable interest to the recent rediscovery of one of the old authentic Indian emerald mines in the South American Andes which was lost for over a century. The facts as to emerald production generally are of interest, especially as the gem has steadily increased in favor during the past thirty years in continental Europe and the United States; it has, as every one knows, been the most prized gem of eastern countries for centuries. The monarchs and rajahs and oriental courts of such nations as India, Persia and Turkey have always furnished a ready market for the superior stones and the less distinguished inhabitants are equally good purchasers of less costly stones.

The "real" emeralds, as distinguished from Siberian stones which are not at all comparable in beauty to the South American gems, are entirely produced from one mining district, called "Muzo," in the republic of Colombia, South America. The gem was mined by the native Indians for centuries previous to the discovery and conquest of the plateau of Bogota in the Andes and the Indians operated three mines widely separated geographically, named "Muzo," "Cosquez" and "Somondoco." The stones were regarded as sacred and were relegated to the use of the rulers, priests, and for ceremonies of various natures—possibly, too, as a national asset.

About 1555, under Capt. Pedro de Valenzuela, the Spanish "conquistadores" took over the mines, enslaved the native Indians and compelled them to work the mines. So eager were the Spaniards to get rich quickly that atrocious cruelties were practiced on the Indian workers and this was carried so far that finally the priests complained to the Crown (King of Spain) that the innumerable

* E.M. Columbia, 1909.

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deaths of the Indians employed in the mines adversely affected the ecclesiastical revenues. This resulted in the importation of African negroes, but eventually the mines were partly closed. During the War of Independence in 1816 and later, the whole region was so desolated that two of the mines, Cosquez and Somondoco, were entirely lost and Muzo has produced all the gems since that time. It has been prolific in its output, although there have been many "lean" years, but the entire output has been steadily declining during the past ten years and, according to the very best information, the ultimate practical abandonment must come in the near future unless new veins are uncovered, which is deemed improbable by the foreign English engineers formerly in charge of the workings.

The lust for exploration and search for lost treasure dwells ever in the heart of every Spaniard and for several years a Colombian named Francisco Restrepo, guided only by a few hints contained in ancient Spanish parchment maps in the government archives in Popoyan, wandered far and wide looking for the lost emerald mines of Somondoco. The proverbial search for a needle in a haystack would have seemed better advised as far as possibility of success was concerned, especially as Señor Restropo knew nothing of geology nor emeralds; yet in 1896 he came upon traces of ancient workings and later uncovered very extensive workings which proved to be the real treasure trove, the lost emerald mine of Somondoco, which gives every promise of duplicating the wonderful record of Muzo which was probably $2,000,000 to $4,000,000 annually for a century, and for unknown centuries in pre-Spanish times.

As the property has been fully confirmed as the bona fide Somondoco of legend and will probably produce a hundred million of gems and possibly vastly more, a brief description follows: The mine is situated upon a sectional ridge of the great eastern range of the Andes Mountains in Colombia; I found the old Spanish ditch, which once conducted water to the mines to be some twelve or fifteen miles long; the reservoirs were on top of the mines and constructed in part of dressed sandstone brought from a distance. The open cuts or quarries were gigantic and tunnels are to be found in different places all over the emerald zone which was found to extend about six miles east and west, and three miles north and

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south. There were some thirty tunnels which in every case had been driven in pursuit of an emerald vein which had out-cropped. Mr. Christopher E. Dixon, formerly engineer in charge of Muzo mines, personally extracted over a pound of emeralds by scratching about in one place and another. The tunnels are generally in excellent preservation after a lapse of more than one hundred years.

Several faces of these mines measure from 700 meters to 300 meters almost perpendicular; the first 100 meters is of emerald-producing formation, the remaining 600 and 200 meters respectively is rock of an earlier period and non-productive. Judging from the workings executed by them, the Spaniards must have operated for over one hundred years. The tunnels were never driven in search of veins, but always drifting on veins actually producing gems.

The emerald-producing zone is covered by forest and jungle and certainly there are many tunnels yet to be uncovered. Each tunnel so far explored penetrated into the mountain side on the average 100 yards. At Muzo only special formations exceed this depth which argues well for Somondoco's extent.

The altitude is 9,000 feet, which in connection with the situation within 100 miles of the equator gives the locality a climate of perpetual late autumn.

Emerald Mines Working.—The mines are worked in a series of huge steps—the almost perpendicular fronts on the mountain side from the top downwards measuring each more or less 250 feet—and are composed of dark grey to black-colored clay-slate and limestone rock in which Cretaceous fossils are somewhat abundant; veins of semi-decomposed quartz traverse in zigzag form the rock formation, and in these veins are found the emeralds, sometimes a few at a time and occasionally in pockets of contorted shapes. Isolated emeralds are found, that is, not nested in the vein but alone in the rock formation and these are generally very prime and fine stones. They are usually lost, however, as they escape unobserved in the ordinary course of excavation and go down into the ravine with the débris.

Provided the mass of the rock is of well-defined stratification, the darker the color of the rock the better. It is difficult to describe to the uninitiated in this class of mining what is termed a "satisfactory formation"—one which produces good quality emeralds.

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About all that can be said is that to be confident of good results the formation should be of black rock measuring about 30 × 30 × 30 yards of horizontal stratification, or slightly inclined but not vertical. After a few yards of surface earth and decomposed rock has been removed the finer rock formation can be easily studied. In some cases where the mountain front is perpendicular a few yards fall away each year, exposing to view the real stratification of the rocks and thus a study of the formation is easily effected—in this case and not when the surface is covered up—should an emerald-bearing vein be anywhere near to the surface.

In time a land-slide or a breaking away of the face will carry away a portion of the vein and the disintegrated emeralds roll down the hill to the creek, where occasionally one or two may be picked up and thus bring about a search for the place where they are likely to have come from.

A formation (productive) should comprise firm or compact rock (not decomposed); the stratification may vary in thickness from two feet to one eighth of an inch—the thinnest laminas invariably produce the finest stones in the veins that traverse it; the rock although firm and well compressed should not be *harsh or rough to the touch but smooth and silky.* The veins should be loose and hollow and, if broad and stony, are likely to produce large stones and large quantities of them. The fine quality gems are not usually embedded or nested in thick, bold veins but in narrow soft quartz. Hard glassy quartz veins are never desirable.

The quality of the emerald, that is, the depth of the color, corresponds with the shade of the rock in which the producing vein is found to exist—as the rock is so is the emerald. The emerald-producing rock is of sedimentary formation. The emerald is produced from the chemical constituents of the rock. The theory is that the silicate of alumina and glucina in liquid state found its way into the veins which formed in the long and short cracks after contraction had taken place. The liquid silicate flowed or trickled down the inclined vein until it found a place of rest where under undisturbed conditions it crystallized into what is termed an emerald—the number of stones in a pocket depends on the amount of liquid silicate, and the purity of the quality (its color) depends on the quality of the rock from which the liquid originally issued.

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The clearness of the stone depends, of course, on the continued condition of freedom from disturbance during this process of crystallization and the almost infinitesimal but constant decrease of temperature.

The more marked a given formation is by anticlinal and synclinal curvatures the better, and if a vein is discovered travelling in the trough of the syncline the production of stones may be immense.

The lost emerald mines of Cosquez still lies unknown in the tropical Andean jungle but Somondoco (sometimes called "Chivor" from the valley near it) will soon yield gems to delight the world.

Would not the diary of Señor Restrepo be interesting reading had he set down each day during his long search his hopes and fears and dreams and his final triumph when the actual gems from lost Somondoco glittered in his hand. The yield of the greatest modern gold fields under the most improved machinery will possibly not equal the annual revenue this actual El Dorado will give Restrepo—the lone searcher.

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THOMSONITE IN NEW JERSEY.

By F. A. CANFIELD.

The occurrence of thomsonite in New Jersey has long been in doubt. The earliest printed record of such an occurrence that has been found by the writer, is in the report of state geologist of New Jersey for the year 1868.*

In the "List of Minerals in New Jersey" it says: "Thomsonite Bergen Hill, Hudson County. In radiated masses like pectolite, but more compact and dull—sometimes of a brown color."

On this authority the writer included this mineral in his "Catalogue of Minerals found in New Jersey."†

A search through ten old collections failed to find more than four specimens labeled "Thomsonite, Bergen Hill, N. J."; three of which are pectolite, the other is natrolite.

In the year 1875 a few specimens of a strange mineral were found in the second tunnel through Bergen Hill.

The mineral was in small white masses implanted upon the surface of prehnite. Under a microscope it appeared in wheat sheaf tufts which stood on the smaller end, and radiated or flared in parallel blades, or crystals, like the leaves of a half-opened book.

The writer found one of these specimens in an old collection, labeled "Epistilbite." On this authority he applied this name to his own specimens.

In the year 1906 the writer obtained two curious specimens from the quarries at Paterson. They were called stilbite. One was a hemisphere 30 mm. in diameter, attached to the rock. The surface was dull but even. It was thought to be prehnite. The other specimen consisted principally of datolite and natrolite, but imbedded in the latter mineral was a cluster of brownish crystals, which under a glass showed the parallel grouping of thin blades, which was so characteristic in the above-described specimens from Bergen Hill. They were finally labeled epistilbite.

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* "Geology of New Jersey," 1868, p. 749.

† "Final Report of the State Geologist" (of N. J.), Vol. II, 1889.

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During the summer of 1910 more specimens were found at Paterson which showed either the globular or the sheaf-shaped structure. They were not thought to be the same mineral until specimens were found which showed a combination of the two forms. The structure of the spheres is radial. In rare cases the blades are prolongations of some of the rays. At first glance the blades suggest stout crystals, but when they are magnified they show a transparent glassy core which is covered with very thin blades (crystals?) which overlap one another. Some of the spheres are 40 mm. in diameter. Simple tests were made, but not being satisfactory, the following analysis was made of some very pure material. It proves the mineral to be thomsonite.

	Analysis	Theoretical Composition when the Ratio of Ca : Na_2=3 : 1.
SiO_2	37.90	37.0
Al_2O_3	31.40	31.4
CaO	12.80	12.9
Na_2O	4.53	4.8
H_2O	13.05	13.9
Loss	.32	
	100.00	100.0

K_2O was not determined; possibly it is represented by a portion of the loss.

It will be noticed that this analysis agrees very closely with the theoretical composition given in Dana's "Mineralogy."*

The tests of the Paterson mineral show that it fuses at 2. It gelatinizes easily with HCl. Hardness is a little more than 5. Density is 2.35.

The color varies from pure white to a delicate gray. The luster is pearly in the balls and vitreous in the blades. Some of the blades are grouped in the form of an hour glass, or a double wheat sheaf. The only good crystals are microscopic. The associated minerals are prehnite, calcite, laumontite, analcite, natrolite, datolite, albite(?), also that mysterious mineral which has disappeared, leaving the moulds or casts of its crystals to puzzle collectors of Paterson and Great Notch specimens.

In conclusion it may be said that the so-called epistilbite from Bergen Hill proves to be thomsonite.

Dover, N. J., January, 1911.

* Dana's "Mineralogy," 6th ed., p. 608.

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THE CHEMISTRY OF RAW SUGAR PRODUCTION.*

By CHARLES A. BROWNE.

Few commodities of commerce have acquired so wide and general an interest as sugar, and this interest we find exemplified in many ways. It is shown best of all perhaps in the large number of books and publications which have been written upon sugar. These books, a collection of which in the Library of Congress in Washington has been classified together in one catalogue, constitute in themselves a large library; there are books upon the history of sugar, upon the economics of sugar production, upon the agriculture of sugar-producing plants, upon the manufacture, technology and refining of sugar, upon the chemistry and analysis of sugar, upon the nutritive value of sugar, upon the tariff laws and bounties regulating the importation and exportation of sugar and so on and so forth. The importance of sugar in the commercial affairs of the world is so great that some economists have used the per capita consumption of sugar as a means of gauging the wealth and prosperity of a nation. It may be interesting to note in passing that the statistics of 1910 show the following consumption of sugar per capita for several different countries during the past year.

	Pounds.
England	86.30
United States	81.60
Denmark	77.75
Switzerland	64.10
Sweden	53.90
Holland	43.53
Germany	43.45
Norway	41.78
France	37.80
Belgium	32.36
Austria-Hungary	25.14
Russia	22.82
Spain	14.20

* Lecture delivered to the third- and fourth-year students in chemistry and chemical engineering, January 18, 1911.

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	Pounds.
Turkey	12.89
Italy	9.33
Greece	7.52
Bulgaria	7.05

The phase of the subject which I have been asked to present to you this morning is the part which chemistry and chemists play in the production of raw sugar. Sugar, *i. e.*, sucrose, although composed of the common elements C, H and O—the two latter in proportion to form water—has not been made synthetically up to the present time. I mention this for we read occasionally in the Sunday newspapers of sensational experiments (paid press notices, no doubt) where steam is injected into an electric furnace containing charcoal at one end and a continuous stream of the purest white sugar flows out at the other. Here is a sample of such sugar, testing 100 per cent. exactly as they claim, and here is an illustrated pamphlet telling all about the process. It is simplicity itself, for they show by formula that $C_{12} + 11H_2O = C_{12}H_{22}O_{11}$, and there you are. The most important feature of this, as of all similar announcements is: "The price of our stock is bound to increase rapidly; do not wait till the stock goes higher but buy now."

The process of making synthetic sugar just described is mentioned simply as an example of the countless number of fakes which are continually springing up in connection with the sugar industry; there are fake powders which applied to the soil will increase the yield of sugar cane or sugar beets; there are fake chemicals for clarifying and fake processes of boiling and crystallizing. So many people have been deceived by putting money into fictitious schemes, that sugar planters and manufacturers have become in some countries ultra-conservative; they refuse to adopt a really good invention when it does come along. It is one of the chief qualifications of the chemist who takes up any branch of chemical technology as a profession to be able to detect quickly the false and good features of the numerous new processes which are brought to his notice; he will increase his usefulness and earning capacity in this way many times over.

Sugar may, at some time, be manufactured synthetically just as indigo is now made; and when that day arrives it may also happen that synthetic sugar may displace the agricultural product just as

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synthetic indigo is displacing the product of the indigo plant. But until the day of synthetic sugar does arrive we must be dependent for our supplies of raw sugar upon some one of the sugar-producing plants among which may be mentioned the sugar cane, the sugar beet, the maple tree, the palm tree and various plants of lesser importance as the sorghum and maize.

The oldest and best known sugar-producing plant is the sugar cane. This plant grows only in tropical and semi-tropical countries; it resembles in many ways the Indian corn, producing a jointed stalk varying from six to ten feet or even more in length.

Those of you who are interested in history will find the history of the sugar cane of exceptional interest. The native home of the cane is India and it is mentioned again and again in the old sacred books of the Hindoos and in ancient Chinese writings centuries B.C. The Greek soldiers of Alexander the Great saw the sugar cane growing in India at the time of his conquest and brought back stories of the wonderful reed which yielded a juice sweeter than honey. The Persians and Arabs carried the cultivation of the sugar cane westward and we find that sugar was both grown and refined in the valleys of the Tigris and Euphrates in the tenth century A.D. during the rule of the Arab caliphs. It is to the Mahommedan Arabs that the glory belongs of giving sugar to the European and the western world. The Crusaders found sugar cane and sugar factories in Syria and Palestine and brought back samples of the product upon their return from the East. Ruins of these old sugar factories are found in Judea at this day. The Saracens introduced the cultivation of sugar cane into Sicily and the Moors into Spain; the Spaniards in their turn carried the sugar cane with them to the new world during their voyages of discovery and colonization; and so the sugar cane was carried from its original home in India throughout the entire tropical and semi-tropical world.

In the long years of its history the sugar cane is probably but little richer in sugar than it was when the soldiers of Alexander first tasted its juice. There is an important reason for this: the sugar cane is propagated almost entirely by planting the stalks, the buds or eyes germinating and producing a new cane resembling the old stalk. It is practically impossible to improve a cultivated plant

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without the agency of seed production; there must be cross-fertilization and some process of controlled selection. The sugar cane produces seed in a feathery plume at the end of the stalk, but the seed had been regarded as sterile until about thirty years ago when it was found that under favorable conditions some of the seeds could be made to germinate; thousands of new canes have resulted from this discovery and these so-called seedling canes are displacing many of the old-established varieties. In the nurseries of tropical countries, where new varieties of sugar cane are bred, the problem of selection devolves almost entirely upon the chemist, who determines the percentage of sugar in the different canes, the purity of the juices and other factors which have to be considered.

The sugar cane, when the crop is ready, is harvested by cutting off the stalk as close to the ground as possible, trimming off the green tops and stripping off the leaves. These and the other agricultural operations of planting, fertilizing and cultivating require a large amount of labor, the expense for which makes up about three fourths of the cost of the raw sugar, the remaining one fourth being due to the expense of manufacture.

The composition of the stalks and the expressed juice of the sugar cane vary considerably. The general range of the different constituents as compiled from analyses made in different countries is given in the following table:

	Whole Cane, Per Cent.	Cane Juice, Per Cent.
Water	67 –75	80 –86
Dry substance	33 –25	20 –14
Fiber (cellulose, etc.)	10 –15	
Sucrose	11 –16	12 –18
Invert sugar	0.5– 1.5	0.5– 2.0
Ash	0.5– 1.0	0.4– 0.8
Nitrogenous substances	0.4– 0.6	0.1– 0.4
Gums, acids, etc.	0.2– 0.5	0.3– 0.6
Wax and fat	0.4	0.2

Individual cases may show variations above or below these figures.

The sugar cane after it is hauled to the factory is first passed through mills to remove the juice. The cane mills are of all kinds and types, and range from the crude ox-driven mills employed in the Philippines and other primitive countries, to the high power, steam-driven hydraulic nine and twelve roller mills employed in

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Cuba, Java, Hawaii, Porto Rico, Louisiana and other countries where the most modern machinery is used. In the best equipped factories the cane is delivered by an endless carrier to huge corrugated crushers, which reduce the stalks to a thick blanket of pulpy fiber, removing at the same time some 50 per cent. to 60 per cent. of the juice. The crushed stalks pass next through a mill of three rollers, where still more of the juice is removed, and then through a second, third and sometimes a fourth set of such rollers, the hydraulic pressure upon the rollers being increased at each mill in order to remove more and more of the juice. It is also customary to wet the pulp with a thin spray of water between the sets of rollers, the water thus soaked up facilitating the removal of the residual sugar by the succeeding rollers. This process of wetting, or maceration, as it is called, is highly important, but requires to be carefully controlled; the water added must of course be afterwards evaporated and the question which the chemist must decide is when the cost of evaporation begins to exceed the value of the extra sugar recovered. Large numbers of treatises have been written upon maceration and scarcely any two authorities are found to agree upon the details of manipulation. The quantity of water used for wetting the fiber is usually about 15 per cent. per 100 parts of normal undiluted juice, although 25 per cent. and more is sometimes used. With 15 per cent. maceration about 90 per cent. of the sugar in the cane is extracted in the juice; with 25 per cent. maceration over 95 per cent. of the sugar may be extracted. The residue of cane fiber as it leaves the last mill contains about 45–50 per cent. moisture and from 3 to 5 per cent. sugar, *i. e.*, from 5 per cent. to 10 per cent. of the original sugar in the cane. This residue of fiber is called "bagasse" and is burned under the boilers; it constitutes the chief and in some countries the only supply of fuel for operating the sugar factory.

The polarization and purity of the raw juice are the first important factors to be determined in the chemical control of a cane sugar factory. The polarization of the juice will give the approximate sugar content; the dissolved solids in the juice are estimated by means of a floating hydrometer called a Brix spindle. The polarization of the juice multiplied by 100 and divided by the reading of the Brix spindle gives the purity of the juice. Good cane

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juices run over 90 per cent. purity, juices running from 85 to 90 per cent. purity are fair, and from 80 to 85 per cent. medium. Juices with a purity below 80 per cent. are poor and very unsatisfactory to work.

The second step in the manufacture of cane sugar is the clarification or purification of the raw juice. The best clarifying agent and the one that has been used from time immemorial is lime. Nothing better has ever been found, although hundreds of patents have been taken out for substitutes.

Many methods of using lime are practiced and a few of these will be described. Cane juice as it comes from the mill is slightly acid. One method of clarification is to neutralize this free acid of the juice by adding lime to slight alkalinity, and then to heat to boiling. The lime combines with the organic acids and phosphoric acid of the juice and the heat coagulates the albumenoid substances; a thick scum of impurities rises to the surface, this is skimmed off and the hot juice run into settling tanks when the suspended impurities settle out. More often the juice is passed through filter presses and the impurities removed in this way. The residue of impurities called "filter press cake" contains the phosphates and nitrogenous matters of the juice and is returned to the cane fields as a fertilizer.

In many factories the cane juice is sulfured before liming; gaseous SO_2, produced by burning sulphur, is led into the juice to a certain point of acidity; the free acid is then neutralized with lime and the juice heated as first described. The SO_2 has a favorable bleaching effect upon the juice and the mechanical separation of the impurities is greatly facilitated by its action. Objections against its employment are the increase in scale (mostly $CaSO_4$) which forms upon the tubes of the evaporators and the contamination of the molasses with sulphites, the latter being forbidden in food products by the laws of some of the states. In some factories phosphoric acid is used with the lime; this gives a beautiful clarification but does not give the bright effect which bleaching with SO_2 brings about.

In some countries, notably in Java, lime is added to the juice to strong alkalinity and the excess of lime then removed with CO_2. This process of clarification called carbonatation is the only one

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used in beet sugar manufacture. It works well with cane juices when but little invert sugar is present. If the latter occurs in large amounts the lime forms dark colored soluble compounds which not only give a dark colored sugar but interfere seriously with the work of evaporation and crystallization. Such juices are said to be lime-burnt. The tendency at present in cane sugar manufacture is against carbonatation and all other methods of strongly alkaline clarification.

The third process in the manufacture of cane sugar is that of *evaporation*. In primitive countries and out-of-the-way plantations evaporation is carried out over the direct fire in open pans or kettles. The juice is either boiled down in one single kettle or passed through a train of pans; when crystallization of the sugar has begun great care must be exercised that the mass be kept in constant motion; otherwise there will be burning and caramelization next to the surface of the evaporator. Such caramelization is in fact unavoidable and all open kettle sugars are characterized by a dark color and by an agreeable aromatic taste which is preferred by many to that of the pure refined sugars. In some countries the cane juice after evaporating to a thick pasty mass is allowed to cool and solidify, just as molasses candy hardens after cooling. This solidified mass is called *concrete sugar* and is ground up in mills and marketed as a coarse lumpy sugar of very uneven composition. This concrete sugar contains of course all the molasses with the soluble impurities of the juice. Such sugar constitutes at present almost the sole output of the Philippine Islands. It is shipped to this country in mats weighing about 50 pounds and comes in three grades which contain all the way from 80 per cent. to about 90 per cent. pure sucrose.

In other primitive countries, especially in parts of South America, the juice is not evaporated to concrete but only to the consistency of a thick mush; this mush is run into hogsheads having a fine perforated bottom through which the molasses or mother liquor surrounding the crystals of sugar percolates. When as much as possible of the molasses has drained away the residue of sugar is removed and sold as *muscovado* sugar. This is purer than concrete sugar and polarizes sometimes as high as 92 per cent. Is it usually quite moist and for this reason very liable to deteriorate.

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In the open kettle process of evaporation there is always considerable loss of sugar due to caramelization and inversion caused by the high temperature of heating which may be from 20 to 30° F. above the boiling point of water. To avoid these losses all modern sugar factories employ vacuum evaporators which allow evaporation to proceed at a temperature much below the boiling point of water and at the same time permit the utilization of waste steam from the exhaust of the engines and other points about the factory. Vacuum evaporators are manufactured in many different forms, and are arranged usually in a series of two, three or four, sometimes even as high as five or six, the combination being called double, triple, quadruple, quintuple, or sextuple effects. In the first vessel of an effect, a lower vacuum is maintained than in the second, a lower in the second than in the third, and so on, the temperature of boiling for each succeeding vessel is thus progressively reduced. The steam which is evaporated from the juice in the first vessel goes to heat the coils of the second vessel, the steam from the second vessel goes to heat the coils of the third, the reduction in temperature of the steam for each vessel being of course counterbalanced by the greater vacuum and lower temperatures necessary for boiling. With a long series of vessels as in a quadruple or quintuple, or sextuple effect, the thin juice in the first body may be boiled under atmospheric pressure or even at a few pounds above this; this is necessary in order to get a high enough temperature to carry sufficient heat through to the last evaporator. The subject of multiple evaporation is a science in itself and exhaustive treatises have been written upon this one single phase of sugar manufacture.

After the clarified juice has been evaporated to a syrup we come to the fourth stage of the process of modern sugar making—the graining or crystallizing of the sugar. This is accomplished in a vacuum pan which is operated in much the same way as one of the vessels of an effect; in the case of the vacuum pan and the same is true with many other effects, the process is assisted by connecting the outlet with a vertical condensing column 28 feet high. A stream of cold water flows through the column and this serves both by rapid condensation of the steam and by the barometric pull of its column of liquid to maintain a high degree of vacuum.

To operate the vacuum pan a charge of syrup is first drawn in;

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this syrup as it leaves the evaporators has a sp. gr. of about 1.25 or about 50 per cent. solids and is boiled down in the vacuum pan to a sp. gr. of 1.50 or about 90 per cent. solids. The ebullition in the vacuum pan is violent and unless the sugar boiler is careful some of the syrup may be carried over with the vapor into the condenser; this is called *entrainment* and is a source of frequent losses in sugar manufacture. In all modern sugar factories the chemist makes constant examination of the condensation water, using the familiar α-naphthol test with H_2SO_4, this test being delicate enough to detect one part of sugar in many thousands of water.

The handling of the vacuum pan requires more skill than any other operation of the sugar house; care must be taken to avoid entrainment and care must be taken to build up crystals of uniform grain or size. The usual practice is to boil down the first charge of syrup to what is called "string proof," *i. e.*, to the point when a few drops of syrup withdrawn from the pan will draw out between the fingers in fine strings or threads. When this point is reached a large charge of fresh cold syrup is drawn into the pan; the sudden cooling of the supersaturated contents starts the formation of innumerable fine crystals. These first crystals constitute the foundation so to speak of all the sugar obtained in a given boiling or strike of the pan; the boiler aims to build up these crystals without forming new ones; he aims from now on to avoid supersaturation and to avoid sudden chilling through drawing in too much syrup at one time. He controls his process by drawing out samples every few minutes and examining these upon glass against a light; if he sees fine new crystals appearing among the old ones he reduces the vacuum a little, thus raising the temperature and dissolving this false grain as the fine crystals are called. By skillful manipulation, which only comes with long practice and experience, the sugar boiler is able to build up his crystals to any desired size; the usual practice is a crystal about the size of ordinary granulated sugar; in certain localities, however, a large crystal is favored, as for example, in Peru, where the sugar is boiled slowly and for a long time, thus building up a very large grain.

When the vacuum pan is filled with a thick magma of sugar crystals, of about the consistency of mortar, the steam is shut off, air is admitted, the bottom of the pan opened, and the entire con-

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tents dumped into a mixer, which keeps the mass in slow movement by means of revolving arms. This mixer is situated over a row of centrifugal machines; the mass of crystals called sometimes *masse cuite* from the French or Füllmass from the German is drawn off gradually in successive charges into the centrifugals. The inner walls of the latter are lined with fine brass meshing and as the drums are rotated the masse cuite is whirled against the meshing which retains the sugar but allows the molasses to pass through. After spinning for a few minutes until as much of the molasses is removed as possible, the revolving mass of sugar may be sprayed with a fine spray of water or a jet of steam in order to remove more of the film of molasses which remains adhering to the crystals; the amount of spraying depends upon the whiteness of sugar desired. In Louisiana a very pure, white sugar is made by spraying with several sprinklings of water; such sugar is over 99 per cent. pure sucrose, the remainder being mostly moisture. In Cuba and Porto Rico they aim to make a 96 per cent. sugar. In Hawaii and Java a sugar testing about 97 per cent. is desired. Spraying will, of course, dissolve some of the sugar, so that the process is one which must be carefully controlled.

After the sugar is spun out, the centrifugals are stopped and the sugar emptied through the bottom of the drum into a conveyor by which it is carried to the bagging department where it is prepared for shipment. The raw sugar from the centrifugal contains considerable moisture and in some countries the sugar is dried in revolving drums before being bagged. This drying is advantageous for two reasons: (1) The excess moisture is removed, thus saving the transportation charges upon water, and (2) the sugar is sterilized and protected against the attacks of ferments and bacteria. The drying of raw sugar is not practiced in Cuba, Porto Rico or Louisiana, but is carried out in Java and the Hawaii Islands where the sugar has to be shipped for long distances to this city and other ports for refining. The storage of undried raw sugar for long periods of time is a risky operation, as many speculators in sugar have found to their cost.

The sugar which is made from the pure juice of the cane is called "first sugar" and the molasses drained from this sugar is called "first molasses." The latter still contains a large amount of

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sucrose and various processes are used to recover as much of this as will crystallize. The first molasses is sometimes boiled down again in the vacuum pan and a second crop of sugar crystals obtained; this is the second sugar and the molasses obtained from this the second molasses. The second molasses may be boiled even again and a third sugar obtained, the molasses from which is the third or final molasses. Of course, as the sugar is removed the impurities become more and more concentrated in the molasses, until finally a thick stringy mass is obtained which will no longer crystallize. Such a molasses may still contain, however, 30 per cent. sucrose; there is also present about 30 per cent. invert sugar, 8–10 per cent. of ash, and 8–10 per cent. of gums, organic acids, amido-compounds, etc. This residual molasses is unfit for human food, although attempts are made to bleach it, dilute it with glucose and make it otherwise presentable. The waste molasses is valuable, however, as a cattle food and is also used by distilleries for making alcohol.

The tendency of modern methods in cane sugar manufacture is against the repeated boiling of molasses; the aim is to get as much sugar as possible in one operation. Many processes have been devised to attain this end. One method is to take the molasses from the first strike of sugar, draw this into the vacuum pan with the syrup for the succeeding strike and boil the two down together. The masse cuite from this mixture is then run while still hot into large tanks, called crystallizers, where it is kept in slow motion by means of revolving arms; as the mass cools and thickens more molasses is drawn in to keep the proper degree of fluidity. When no more sugar will crystallize as determined by analysis of samples, the contents of the crystallizer are spun out in centrifugals and the molasses withdrawn from the factory. There are many modifications of this process, each sugar manufacturer having usually a pet scheme or method of his own.

With this very hasty description of the sugar cane industry we will pass to the great rival of the cane, the sugar beet. The history of the sugar beet industry extends back only about one hundred years, but though centuries younger than the cane industry the story of the development of beet sugar manufacture is in many respects more interesting to the chemist. The sugar beet may be

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called the foster child of chemistry; in its wild uncultivated form, even as we find it growing to-day, the primitive sugar beet is very deficient in sugar. The beet, however, belongs to a much younger family of plants than the cane; its habits are not so firmly fixed; it accommodates itself more easily to new conditions. The propagation of the beet by seeds, instead of by buds as with the cane, has rendered it easy to build up the sugar-producing capacity of the plant until it has reached and in some cases even passed beyond that of the cane.

I ought to mention here the important historical event which led to the founding of the beet sugar industry; it was the great continental blockade established by the fleets of England against the whole continent of Europe during the wars with Napoleon. The nations of Europe shut out from their supplies of cane sugar were driven to devise a substitute; in some countries as in Bohemia, they made sugar from the maple, in other countries they crystallized dextrose from the juice of the grape, but these sources were inadequate. The final outcome of the matter was the birth of the beet sugar industry. It was one hundred years ago in 1810 that the first loaf of beet sugar was made and presented to Napoleon; and the centenary of this event has just been celebrated in France.

It is doubtful if the beet sugar industry could have made much progress, had an important piece of apparatus, the polariscope, not been invented about this same time. The polariscope offered a quick and rapid method for determining the amount of sugar in beets. There was thus established a means of selecting beets for the production of seed—mother beets as they are called. A small boring is taken diagonally through the heart of the beet and the percentage of sugar determined in this by the polariscope. The beets of highest sugar content are planted for seed; the same process is carried out with the second generation of beets grown from the seed of the first and so on generation after generation. In this way the original sugar content of the beet of only a few per cent. has been raised to 14, 16 and even 18 per cent. The process of seed selection is still kept up and in the seed nurseries chemists are kept constantly employed polarizing mother beets for seed production. The work demands constant vigilance; a plant easy to improve will deteriorate even more easily; if the sugar beet were

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left to itself it would revert in a few generations to the old condition of low sugar content.

Extremely large beets are usually deficient in sugar and are not desirable; a medium sized beet is most preferred. Many beet growers place considerable stress upon the spiral twist which the root sometimes assumes; this factor, however, is probably of little significance.

Before taking up the details of manufacture we should look for a moment at the composition of the sugar beet and its juice. The average range in composition is given in the following table:

	Sugar Beet. Per Cent.	Sugar Beet Juice. Per Cent.
Water	75 –85	78 –84
Dry substance	15 –25	16 –22
Fiber (cellulose, etc.)	4 – 6	
Sucrose	12 –16	13 –17
Invert suger	0.0– 0.3	0.0– 0.3
Ash (salts)	0.8– 1.5	0.6– 1.0
Nitrogenous substances	1.5– 2.5	0.8– 1.5
Gums, acids, etc	0.4– 0.8	0.3– 0.6
Wax, fat, etc	0.2	

It will be noted as compared with the cane there is much less fiber in the beet and more water; there is also more ash (or salts) and more nitrogenous matter, but much less invert sugar than in the cane. These differences in composition have an important bearing upon the differences in process of manufacture.

In the first beginnings of the beet sugar industry the beets were crushed and the juice pressed out in much the same manner as with the cane; this process, however, soon gave place to the process of diffusion. The beets, after they are dug and deprived of their green tops, are hauled to the factory; they are first washed to remove adhering dirt and then passed through revolving knives which reduce them to fine slicings or chips.

The fine slicings are next carried by a conveyer to the diffusion battery, which consists of a series of tall boiler-shaped cylinders called cells. These cells are connected together by pipes, the outlet from the top of one cell passing downward into the bottom of the next and so on around. Each cell is filled with beet slicings through a man-hole at the top and when full is tightly closed with a cover which is clamped into place. Twelve cells connected in

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series usually constitute a battery and when all are filled, warm water of about 80° C. is passed through the system. The water circulating through each cell removes the sugar from the beet slicings and of course becomes richer and richer in sugar with each succeeding cell. Heaters are placed between the cells so that the circulating water is kept always at the right temperature. When the water has made a complete circuit through the twelve cells of the battery the slicings in the first cell are practically exhausted; this cell is then thrown out of circulation, emptied of exhausted chips, refilled with fresh slicings and reconnected with the system, while the second cell undergoes replenishing. The process is thus a continuous one; ten cells are always in circulation while one is always being emptied and one always being refilled. This goes on continually during the beet campaign, night and day, Sundays and holidays; interruption at the diffusion battery upsets the work of the whole factory.

The exhausted slicings from the diffusion cells are dried by the heat of the flue gases from the boilers and are then sold as a cattle food.

The diffusion juice as it leaves the last cell of the battery contains from 12–15 per cent. sugar and is then ready for clarification. The juice is first treated with a considerable excess of lime; and the dissolved lime precipitated by leading in a stream of CO_2. This process, called "carbonatation," has already been described.

The burning of lime is an important operation in every beet sugar factory, the lime being used for saturating and making saccharate; and the kiln gases, which contain on the average about 30 per cent. CO_2 being pumped off for carbonating.

After the first treatment with lime and CO_2 the precipitated $CaCO_3$ and other impurities are filtered off in filter presses and the juice subjected to a second carbonatation at a higher temperature. Less lime is used in this second carbonatation, the final alkalinity being only a few hundreds of a per cent. CaO.

The juice from the second carbonatation is again filtered, when it is evaporated, grained, and centrifugalled, these processes being carried out exactly as described for cane juice.

There is a great difference in the physical properties of raw cane sugar and raw beet sugar. Raw cane sugar has usually a pleasant

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fragrant odor and a delightful taste which many prefer to the refined product; this is due to the presence of invert sugar and the slight caramelization which this invert sugar has undergone during manufacture. Raw beet sugar, on the other hand, has a very objectionable odor, due to the nitrogenous substances of the beet root and a very bitter taste due to the mineral salts taken up by the beet root from the soil. Beet sugar is not a fit article of food until it has been refined. Attempts have been made to render raw beet products more palatable by subjecting them to a slight inversion and caramelization but this method has been only partially successful.

Many processes have been devised for recovering the sugar contained in low grade beet molasses. In the short time at our disposal we must pass over the osmose process, barium process and other methods of desaccharification and describe briefly the saccharate process of Steffens. In this process the molasses is diluted with water to about 12–14 per cent. solids and the solution treated in the cold with very finely powdered lime, using all the way from 80–150 parts of lime to 100 parts of sugar. Constant agitation of the solution is necessary to prevent the formation of clumps of lime and care must be taken to keep the temperature down. The insoluble tri-saccharate of lime which is formed is filtered in presses and washed with cold water to remove impurities. This tri-saccharate is then mixed with hot washings from the filter presses and added directly to the diffusion juice for saturating; the saccharate is decomposed by the CO_2 and the liberated sucrose worked up with that of the juice. In some factories the saccharate sugar is treated separately; there are many modifications of this process and nearly every factory has a method of its own.

A peculiar impurity found in sugar beet products is the sugar raffinose, which has a polarizing power over 50 per cent. greater than sucrose. A very small percentage of raffinose will introduce serious errors in the control of factory operations and the chemist may wonder why his yield of crystallizable sugar does not come up to his polarizations. The sugar refiners in England have been much agitated the past year by finding large amounts of raffinose in raw beet sugars from Germany and a congress was held in Berlin to arbitrate the question.

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In addition to raffinose there are gums and pectinous substances in the beet which affect the polarization and introduce errors in the calculation of the sucrose balance. Similar gums are also found in cane products. Every well-organized cane sugar and beet sugar factory maintains a strict account of all the sugar which enters the factory in the cane or beet and all the sugar which leaves the factory in raw sugar or molasses. The difference between the two should equal the determined losses of manufacture, such as sugar lost in the press cake, sugar lost in the spent beet chips, etc. If there are any undetermined losses the chemist is called upon to explain; the papers and discussions upon undetermined losses would alone fill several volumes every year.

In the present condition of the sugar industry the chemist has to understand not simply the chemistry of sugar; he must also be perfectly familiar with the chemical properties and behavior of all the non-sugars which do and may occur in his raw products. A German chemist, Rümpler, has written a book upon the non-sugars of the beet; the list of compounds runs up into the hundreds. Every technical sugar chemist should master the contents of this book. It is a pity that no such work has been written upon the non-sugars of the cane.

The application of strict scientific principles has been followed more energetically in beet sugar manufacture than in cane sugar manufacture and this we would expect for the beet industry was developed in the highly civilized European countries where skilled labor was abundant and the cane industry was maintained in primitive tropical countries where labor was shiftless and unskilled. This condition of affairs existed for many years and gave beet sugar for over two decades a strong ascendancy over the cane. In 1852, taking the whole supply of the world, there was six times as much cane sugar manufactured as beet sugar; beet sugar kept gaining, however, until 1884 the production of the two kinds of sugar was about equal, a little over two and one half million tons each. Beet sugar then took the lead until in 1899 the production of beet sugar was about five and one half million tons and that of cane about three millions. Then came the ending of the Cuban war and with it the introduction of the scientific principles worked out for the beet industry to the old dilapidated cane factories of

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Cuba and Porto Rico. Modern methods of manufacture had also been busy reorganizing the cane industry in Hawaii, Java, Australia and other countries. The production of beet sugar in 1901 reached nearly seven million tons and since then has remained about stationary; but the increase in cane sugar production has gone forward each year by leaps and bounds. In 1907 the cane had caught up to the beet, the production being a little over seven million tons apiece. In 1908 cane sugar was over 600,000 tons ahead and in 1909 nearly 2,000,000 tons ahead out of a total world's production of about 15,000,000 tons. This supremacy the cane seems destined to maintain.

There are several reasons for the belief that the future development of the sugar industry will be more along the cane lines. There is first the cheaper cost of producing cane sugar; the average cost of producing one pound of raw sugar being about 1.75 cents for the cane and 2.25 cents for the beet, a difference of 0.5 cent in favor of the cane. There is second, the much greater yield of sugar per acre which the cane gives over the beet. Under the best systems of cultivation and manufacture such as are found in Hawaii and Java, over four tons of sugar can be produced per acre. Germany, which leads in the production of beet sugar and in the yield of this per acre, can raise barely two tons of sugar per acre. Then there is third, the preference which the refineries of England give to cane over beet sugar. Yet notwithstanding these advantages possessed by the cane industry, the production of beet sugar seems destined to hold its own for many years to come; and indications are that the present year will see a record-breaking beet sugar crop of 8,500,000 million tons. The consumption of beet sugar, however, will be limited largely to those countries where it is produced; it cannot compete with cane sugar in foreign markets.

In the following table I have arranged some twenty leading sugar-producing countries of the world in the order of their production for the year 1909–10.

	Tons.	
1. Germany	2,027,000	beet.
2. Cuba	1,804,349	cane.
3. Austria	1,257,000	beet.
4. Java	1,200,618	cane.
5. Russia	1,145,000	beet.
6. France	801,000	beet.

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	Tons.	
7. Hawaii	462,613	cane.
8. United States	450,595	beet.
9. United States	335,000	cane.
10. Porto Rico	308,000	cane.
11. Brazil	253,000	cane.
12. Belgium	250,000	beet.
13. Mauritius	244,597	cane.
14. Holland	198,000	beet.
15. Mexico	160,000	cane.
16. Formosa	160,000	cane.
17. Peru	150,000	cane.
18. Australia	134,000	cane.
19. Argentine	125,000	cane.
20. Philippine Islands	120,000	cane.
21. Demerara	101,843	cane.

In both the beet and cane industry the complete utilization of by-products will be the determining factor in the matter of profits. The utilization of cane fiber for paper making, the extraction of cane-wax from the sugar house residues, the manufacture of potash salts, ammonia, cyanide and other chemicals from spent beet molasses, improvements in the manufacture of cattle foods from waste molasses and exhausted chips, utilization of press cake for fertilizers, for burning cement and in other ways, improvements in desaccharifying molasses, manufacture of denatured alcohol from waste molasses; these are a few of the lines along which cane sugar and beet sugar chemists are now working to strengthen their respective industries and place them upon a better financial basis.

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SUGAR REFINING.

By W. D. HORNE.

Just as sugar made from the cane appears first to have been used in the orient and to have followed a westerly path, so we find the art of refining had its origin in the east and progressively traveled westward. Although the sugar cane was in use in India as early as the fifth century B.C., we learn of refining having been practiced first about 500 A.D. in Mesopotamia. A large trade in refined sugar gradually developed between the orient and Europe and in 1470 a certain Venetian was awarded a government payment of 100,000 crowns for discovering a process of refining sugar, which was from that time extensively carried on in Venice. In the sixteenth century Antwerp led in the sugar trade in Europe and in the refining industry. England began refining sugar in 1544 and gradually gained the supremacy over the other European countries in the industry. With the development of refining in Europe, the price fell from two shillings per pound in the thirteenth century to four pence per pound in the sixteenth century. This undoubtedly was partly due to the general introduction of the sugar cane into the West Indies from 1510 to 1650, and the consequent enormous production of sugar and its ever growing commerce.

As early as 1689 there was a refinery in New York City, and by 1795, New York, Philadelphia and Boston refined about 1,200,000 pounds of sugar yearly out of the 60,000,000 pounds which was annually consumed, being about two per cent. of the entire amount. Refined sugar was heavily protected by duty and sold at 20 cents per pound. By 1860 there were forty-one refineries in the United States, producing $42,000,000 worth of sugar annually. Now there are about half as many refineries, turning out about three million tons of sugar annually, worth about $300,000,000.

In 1747 Margraf succeeded in obtaining 6.2 per cent. of sugar from beets and Achard established the first beet sugar factory in Austria in 1769. Napoleon, to encourage the industry on account of the difficulty in obtaining sugar because of the blockade of the

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French ports during the wars, heavily subsidized the industry in France about the beginning of the last century. It gradually spread over Europe. The first experiments in beet sugar manufacture in the United States were made in 1830 and again some beet sugar was made in California, Illinois and Wisconsin during the period from 1863 to 1876. Claus Spreckles built a factory in 1870 at Alvarado, Cal., which was the first to meet with pronounced success. The industry has developed considerably since then, until there are now sixty-four factories, located principally in the western states and Michigan, with a total output last year of 457,000 tons, being 13.92 per cent. of the consumption of the country. By 1900 the production of beet sugar had so increased that it had grown to be 60 per cent. of the world's sugar supply.

In 1910–11 the world's sugar supply will be 17,000,000 tons—50 per cent. cane and 50 per cent. beet. Sugar is produced also in small quantities from the sap of the date palm in some eastern countries. Prolonged attempts were made in this country some fifteen or twenty years ago to produce sugar from sorghum, but without success, as the dextrinous bodies in the sorghum juice prevent large quantities of the sucrose from crystallizing, and this plant is now used exclusively for the manufacture of syrup. The annual production of maple sugar in this country amounts to 6,000 tons, but as maple sugar owes its principal value to its pleasant flavor, rather than to its sweetness, it is always sold in its raw state, as refining it would rob it of half its present value and render it impossible of competition with cane or beet sugar. Milk sugar is produced to some extent and used, always refined, for medical purposes principally.

Sugar refining consists essentially in the purification of the crude crystalline material through the well known process of re-crystallization. To this we can add the second process of absorption of impurities by means of bone-black, first introduced by Derosne in 1812. By this means is affected a direct separation of impurities from the sugar solution and a large removal of the coloring matters, allowing of the more easy production of perfectly white crystals. While the process is relatively a simple one, and perhaps more mechanical than chemical in its nature, the enormous consumption of sugar in the country and keen competition in manufacture have

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led to the establishment of very large and elaborately equipped plants for the conduct of operations where every economy of procedure is carefully observed. The great amount of capital involved in erection and maintenance of such plants, together with the highly developed technique required for their operation has tended to keep the business in relatively few hands and to render it a dangerous enterprise for any but the most thoroughly equipped to venture upon. The sugar refineries of the country are in a few spots—New York, Philadelphia, Boston, New Orleans, San Francisco and Baltimore containing practically all of them. In 1910 the country consumed 3,350,355 long tons of sugar, practically all being refined. Of this amount 333,006 tons were cane sugar raised in Louisiana and neighboring states, 457,000 tons were beet sugar raised and refined during the process of manufacture in our western factories, 2,472,756 tons came from our insular possessions and Cuba, and 72,393 tons came from other foreign countries. We imported practically no beet sugar and we exported practically no sugar at all. With the exception of Great Britain, the United States leads the world in the consumption percaput of sugar, this being 81.6 pounds per person. Most of this goes into direct use for household purposes and yet enormous quantities are consumed by manufacturers of confections, preserves, condensed milk, as well as by canners of fruit, ice cream makers, soda water fountains and many other users.

As will be seen, the natural location for a sugar refinery is on the water front where it can more economically receive and unload the great cargoes of sugar and of coal which come to it almost daily, for a modern refinery melts from a million to three or four million pounds of sugar per day. The economical handling of this vast amount of raw material and the successful guidance of so much organic matter in solution at high temperatures through long and complicated processes, without fermentation, decomposition, discoloring or waste is no mean accomplishment; but as in every other instance of manufacture the whole matter has been worked into a routine, which once properly installed, almost takes care of itself. As the sugar, whether in solution or later in granular form, has to pass through many operations in a continuous stream as it were, it is thus found advantageous to have the units of a sugar

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refinery many stories high, so as to take advantage of gravity to pass these solutions from place to place or in delivering the dryer material from one department to another. This holds true equally of the bone-black, and we find the char house is usually a tall building in close proximity to the sugar house itself. In many instances the boiler house also has its coal bunkers in the upper story, allowing the fuel to feed by gravitation into the fire room or even upon the grates themselves. The high cost of real estate in cities also makes it advisable to extend upward rather than laterally. Each sugar refining plant will be found to have a large dock department for the receiving, weighing and storage of raw sugar, a boiler house for the generation of steam for power and evaporation, a wash house for the so-called washing of raw sugars, the pan house for boiling, granulating and other processes of manufacture of dry sugars, a char house for the accommodation of the bone-black filters, kilns for the re-burning of the char, etc., and usually a warehouse for the storage and shipment of refined products. These departments are not always built separately and frequently more than one of them will be found in the same building.

Somewhat different processes are used for the refining of beet and of cane sugar, but the difference consists principally in the method of defecation or clarifying the original raw sugar solution of its suspended impurities and part of its coloring matter. In either case the first operation is the unloading, weighing and storage of the raw sugar, whether in bags, baskets, mats, hogsheads or in other packages, in proximity to the melting department. A portion of the sugar is usually taken direct from the scales to the wash house, where the sugar is raised by a bucket elevator to an upper story and mixed with a low grade syrup while passing through a conveyor to the raw sugar mixer. The magma of raw sugar and syrup is fed from the mixing tank containing revolving arms into centrifugal machines, which purge the syrup from the sugar in a few minutes. Water sprinkled on effects the final washing, giving a sugar of high purity and a low syrup. This sugar is dissolved in a round melting tank with revolving arms in hot water to a density of about 30 Beaumé, pumped into blow-ups, treated with a very small amount of acid calcium phosphate and made slightly alkaline

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with milk of lime. Such a solution would boil at about 217.5° F. and its temperature is next raised to something under 200° F. whereby tricalcic phosphate is precipitated, entangling the suspended impurities and also removing, possibly by precipitation, some of the coloring matter. The lime has the effect of precipitating some of the gums and the heat at the relatively slight alkalinity precipitates most of the albuminoids. This process is applicable to cane or beet sugar or to a mixture, but in defecating beet sugar alone it is sometimes customary to add a few tenths of a per cent. of caustic lime, precipitating this with carbon dioxide which precipitates suspended matters and largely removes the color. In either case the solution must be mechanically filtered. This is commonly done through cotton twill bags enclosed in woven sheaths. These are called Taylor filters. In the case of beet solutions the filter press is sometimes used, but this is not applicable to cane sugars which are gummy and sticky and clot the filter-press. Beet sugar solutions are sometimes treated with sulphur dioxide after this preliminary alkaline defecation and again filtered in presses in a very faintly alkaline or neutral condition. The filter bags after becoming nearly exhausted are allowed to drain during several hours, are washed with hot water several times allowing to drain between times, and are removed from the closed iron tanks in which they hang, for washing. They are turned inside out and rinsed in several waters, passing through wringers between them. The muddy water thus obtained still contains some sugar and is pumped through filter presses, the sugar of the clear filtrate being recovered. The press cake, containing very little sugar, is discarded.

The clarified solutions are next filtered through bone-black, contained in cylindrical filters or cisterns. The liquor gives up the greater part of its color and a less per cent. of its ash and organic impurities to the bone-black and is collected in storage tanks according to color and purity. Lower grade solutions of greater color follow the washed sugar solution on the char, so that the last filtrate from the char is pretty dark in color and much lower in purity.

The char filtered liquors pass to the vacuum pan, holding about 1,000 to 2,000 cubic feet, where it is boiled to grain and concentrated to a low water content.

This magma is dropped into the mixers or crystallizers, from

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which it passes to the centrifugal machines, where it is purged and washed with a spray of clear water, sometimes followed by a spray of blue water, formerly colored by ultramarine, which of late years has been replaced by harmless aniline colors prescribed by the government, since the pure food law put its ban on ultramarine as a mineral substance, not to be allowed in sugar products. The syrups can again be boiled in the vacuum pan to produce granulated sugar, and when the impurities and color accumulate enough the syrups are used for yellow sugars.

The moist sugar from the centrifugal machines, containing a small per cent. of water, is passed through nearly horizontal revolving drums containing longitudinal shelves projecting inward which have the effect of picking up the sugar, and sprinkling it through the current of warm air which is drawn through in the opposite direction.

The sugar thus dried, next passes through revolving screens which separate it into different grades according to the size of the crystals, giving rise to granulated sugar, to be bagged or barrelled.

In making cube sugar, some of this moist granulated sugar from the centrifugal is pressed into cubical blocks by an ingenious machine, and gently dried in ovens during a few hours. Cut cube sugar was originally made by draining the magma or boiled mass of sugar in conical moulds for about two weeks, with occasional washing by means of pure sugar solution, sawing the dried cones into discs and cutting these across into cubes. A modified process of this kind is in use in Europe and to some slight extent in this country, in which the conical moulds are supplanted by rectangular frames, which are purged in centrifugal machines in minutes instead of weeks.

Yellow sugars are made from low testing syrups, which are boiled in a vacuum pan and contain smaller softer crystals than higher grades of sugar, with considerable amounts of adherent mother-liquor.

To return to the bone-black, this is washed down by hot water after the last sugar solution sinks below its surface and is thus freed from the sugar. Certain mixing of the water with the sugar solution is inevitable, giving rise to a zone of sweet water. As the water begins immediately to dissolve the impurities which the char

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had just absorbed from the impure sugar solution passed through it, this sweet water contains impurities along with the sugar and has to be separated from the main filtrate and treated by itself. The sugar washes out faster than the impurities, however, and when most of the sugar has been removed the stream of outgoing water is turned to the sewer, and continued until the char is pretty nearly free of what water can remove. The bone-black is then drained, emptied from the filter by gravity upon a dryer on the floor beneath, through which it passes to the kilns. These are furnaces provided with internal iron pipes, through which the char passes. By proper regulation of the flow of the char through these retorts its temperature is kept at a point which subjects the impurities still remaining in it to destructive distillation. Moisture, ammonia, carbon dioxide and other gases are given off, some of which are highly inflammable, leaving in the char a small residual amount of carbon. The char after cooling is then ready for use again and this entire cycle of operations is repeated.

When the carbon accumulates to such an extent as to choke the pores of the bone-black, the decolorizing power of the char diminishes, so that finally it must be discarded. Of late years this trouble has been obviated by the introduction of the Weinrich system of de-carbonization in which the char is passed through a roaster very much like the granulator before described and heated on the outside by direct fire to a suitable temperature. The air admitted suffices to burn off the impurities without attacking the carbon of the char itself in any marked degree. By this means char can be kept at a high point of efficiency for a considerably longer time.

The large amount of heat passing away from the char filters in the hot wash water is recovered by passing this water through tubulated heat economizers in which the incoming city water passes in the opposite direction, absorbing the heat of the outgoing water.

Water for sugar refining should be as soft as possible and as free from sulphates, iron, color and suspended matter as can be.

The sweet-waters from the char are evaporated in multiple effect evaporators and usually mixed with other low grade solutions.

Fuel is an important matter in sugar refining, large quantities being used to generate steam for power purposes, evaporation and

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heating. This in large part explains why sugar is not refined where the raw product is made, the fuel cost being prohibitive.

The laboratory organization in a sugar refinery is a matter of great importance. It is the laboratory's function to keep track of the quality, not only of the raw and finished material but also of the material in process of refining, not only to show the efficiency of each process, but to enable the most economical and expedient combination of the various liquors, syrups and other solutions throughout the process of refining so as to prevent unnecessary work and to indicate the most advisable method of handling. All raw sugars have to be analyzed in considerable detail, a close watch is kept on fuel, variations in the water supply must be noted, finished products must be kept up to the mark and waste products must be carefully checked up to avoid excessive loss. New processes have to be devised to meet varying condition and experimental work must always be pursued to keep up with the latest improvements and discoveries in the art.

The routine must go on day and night without cessation and all chemicals, apparatus and other appliances must be properly standardized and those standards maintained. Analytical processes have to be investigated and tests devised for new needs. Many materials entering into use in a large factory have also to be subjected to occasional tests or analyses and competent advice has to be given as to the desirability of new methods or processes, suggested from whatever source.

All this calls for an elaborate, competent and smooth running chemical organization. The routine tests, principally of purities, are conducted by young men carefully trained in this work and provided with every possible facility for its accurate and speedy accomplishment. In the laboratory many tests are made on char samples and wash water daily, all raw materials and waste products, all miscellaneous sugar samples, as those involved in stock taking, and so on. The laboratory also keeps careful statistical records of all its various work and compiles elaborate monthly and yearly statements covering all these tests. The laboratory should be provided further with a good general chemical library and ample files of sugar journals as well as those of more general nature. A good

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laboratory should lead the work of the house, rather than follow it, by showing the way over difficulties that are from time to time bound to spring up; by inventing and elaborating new processes of more economical work; by keeping in touch with the outside world, and in general by assisting a smooth running of the entire enterprise.

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SOME NOTES ON THE LEAD, TIN, ANTIMONY ALLOYS.

By W. CAMPBELL and F. C. ELDER.

The ternary alloys of lead, tin and antimony are of considerable importance in the arts, their principal use being for bearing metals. Other uses might be mentioned such as for type-metal of various qualities and properties, white metal for small castings and so forth.

A careful search through the literature of alloys showed that whilst the antimony kept within a range of not over 25 per cent. the tin and lead varied very widely. In regard to the composition of alloys used in practice the list given in Table I. embraces the most important. A few alloys containing a small amount of copper are also given for comparison. Looking down the list we see that the antimony never exceeds 25 per cent. The lead varies from 93 down to 5 per cent. and the tin from 1 up to 75 per cent. There is no uniform variation however.

The physical properties of this series of alloys are not very well known, at least they have not been published in detail and whilst the anti-frictional qualities within certain limits of use do not show the wide differences one would expect from the variation in composition, there is probably a fairly marked difference in crushing strength as we pass from the lead-rich to the tin-rich side, judging from the structure.

Charpy in a study of white alloys called anti-friction* examined some alloys of the lead, tin, antimony group in regard to their crushing strength. The specimens, 15 mm. high and 10 mm. square, were loaded till certain compressions occurred as shown in Table II. Figures were also given for the binary alloys PbSn, PbSb, SbSn.

Although of such importance it appears that the constitution of the ternary system has not been worked out in detail. Guillet in his "Alliages Metalliques," p. 870, describes four photographs and

* *Metallographist*, II., 41.

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TABLE I.

Name.	Pb.	Sn.	Sb.	Authority.
Electrotype.	93	3	4	Private Notes.
Bearing.	86	1	13	" "
Linotype.	85	3	12	" "
Stereotype.	82	6	12	" "
Bearing.	82	2	16	" "
Stereotype.	82	3.2	14 8	Roberts-Austen.
Bearing.	80	12	8	Guillet. Compagnie de l'Est. Charpy.
Bearing.	80	5	15	Guillet (Guillemin).
Bearing.	80.5	4.5	14.5	As 0.5. Glyco very similar.
Bearing.	78	6	16	Magnolia.
Bearing.	77.7	5.9	16.8	Law. Thurston. Magnolia and Tandem Metals.
Type.	77.5	6.5	16	Private Notes.
Bearing.	77	10	12.5	Cu 0.5. Antifriction.
Bearing.	77	8	14	Cu 1.0. Coleco very similar.
Bearing.	76	14	10	Law. Guillet. Compagnie d'Orleans. Metallic packing.
Bearing.	73.5	8	18.5	Babbitt. American R. R.
Bearing.	73	12	15	Law. Guillet. Compagnie du Nord. Piston packing.
Bearing.	70	20	10	Law. Metallic packing. French R. R. (Charpy).
Stereotype.	70	13	17	Thurston. Mackensie Metal.
Bearing.	70	10	20	Guillet. Paris-Lyon-Mediterranée R. R.
Type.	70	10	18	Cu 2.0. Roberts-Austen.
Bearing.	68	21	11	Babbitt. American R. R.
Bearing.	68	15	17	Law. Guillet. (Dudley. Graphite Metal.)
Stereotype.	67	17	16	Thurston.
Type.	63.2	12	24	Cu 0.8. Private Notes.
Bearing.	62.5	26.2	10	Cu 1.25. Private Notes.
Bearing.	62	27	11	Babbitt Metal.
Type.	60.5	4.5	24.2	Cu 0.75.
——	60	35	5	Private Notes.
Bearing.	60	20	20	Brannt. Roberts-Austen.
Solder.	60	39	1	Private Notes.
——	55.5	40	14.5	" "
Bearing.	48	40	10	Cu 2. Private Notes.
Bearing.	46	36.5	16.5	Cu 1. Babbitt "No. 2." American R. R.
——	42	46	12	Hiorns. Guillet. Hoyle's Metal.
Bearing.	42	42	16	Guillet. Chemins de fer de l'État Français.
Bearing.	40	45	15	Guillet (Ledebur).
Bearing.	40	42	16	Cu 2. German.
Bearing.	37	38	25	Guillet. Italian R. R. (Thurston).
Stereotype.	35	60	5	Thurston (Berthier).
——	33	53	10.6	Cu 2.4. Zn 1. Roberts Austen, White Metal.
Bearing.	10	75	15	Private Notes.
Bearing.	10	75	12	Cu 3. Private Notes.
——	5	75	20	Used for Small Castings.

TABLE II.

Pb.	Sn.	Sb.	Load Corresponding to a Compression of 0.2 mm.	Load Corresponding to a Compression of 7.5 mm.
80	10	10	800 kilos.	1775 kilos.
60	20	20	1050 "	1700 "
40	40	20	1150 "	1825 "
20	60	20	1350 "	2200 "
10	80	10	1100 "	2700 "

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refers to Charpy who concluded that "a microscopical examination reveals the presence in all alloys containing at least 10 per cent. of antimony, of hard grains embedded in a eutectic alloy composed of layers alternately hard and soft. The appearance of these grains remains nearly the same when the proportions of the lead and tin vary, but their hardness changes. In alloys of lead and antimony these grains are composed of pure antimony, in alloys of tin and antimony they are composed of the compound SnSb, and it is probable that in ternary alloys they are made up of a solid solution of SnSb and Sb."

Before taking up the ternary alloys a few notes on the binary alloys will be given.

Lead-Antimony.

Antimony freezes at 631° C., lead at 327° C. The addition of lead to antimony or of antimony to lead lowers the freezing point. The freezing-point diagram showing the relation of percentage composition to temperature of freezing was worked out by Roland-Gosselin* who found that the eutectic or alloy with the lowest freezing-point contained 13 per cent. Sb and froze at 228° C.

Stead† in a paper on the microchemical examination of lead-antimony alloys showed the eutectic contained 12.8 per cent. antimony and froze at 247° C. Gonterman‡ gave 13 per cent. and 245° C.

The freezing-point or thermal diagram is shown in Fig. 1. Temperatures are represented in the vertical scale and percentage composition in the horizontal. *A* or 631° C. shows the freezing-point of antimony which is lowered along the curve *Ar* as the proportion of lead in the alloy increases. Similarly *B* or 327° C. is the freezing-point of lead which is depressed along the curve *Br* as the antimony in the alloy increases. These two curves indicating the freezing out of the antimony and the lead respectively, intersect at the point *r* or 13 per cent. Sb at 247° C., the eutectic alloy which is composed of a mechanical mixture of lead and antimony due to their simultaneous freezing.

* Gautier, *Bull. de la Soc. d'Enc. pour l'Ind. Nat.*, 1896.

† *Journal Soc. Chem. Industry*, March and June, 1897; *Metallographist*, I., 179.

‡ *Zeits. anorg. Chem.*, 55, 419.

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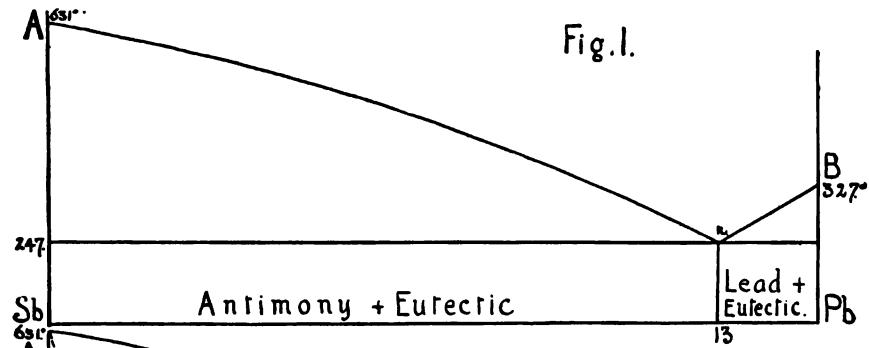

Fig. 1.

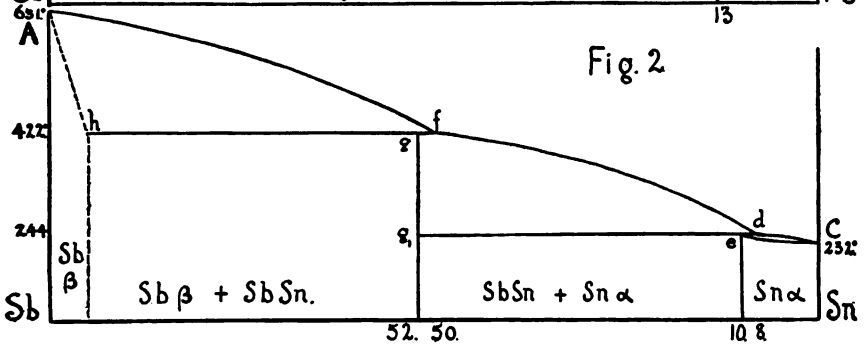

Fig. 2.

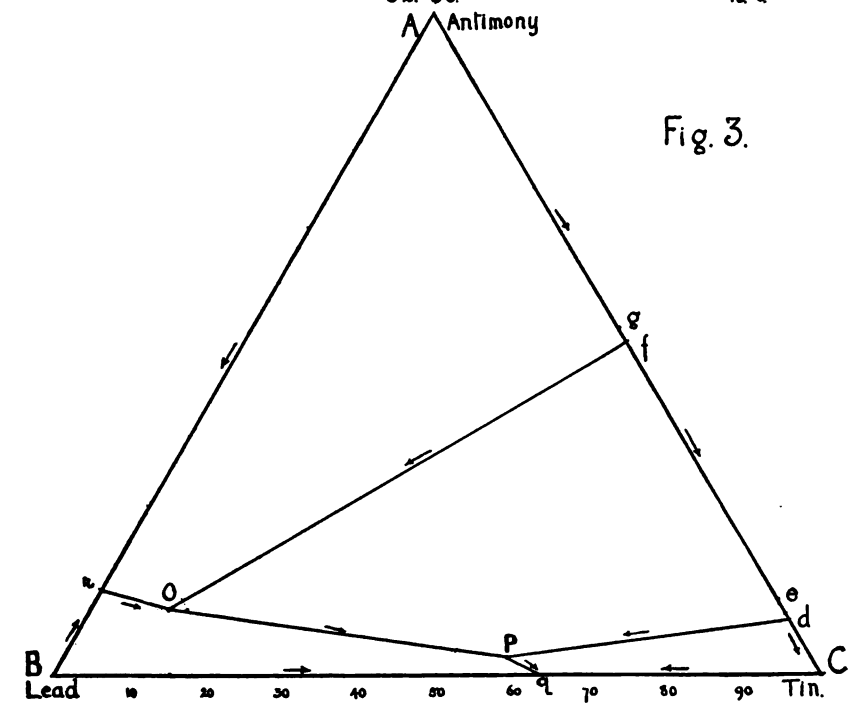

Fig. 3.

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Thus alloys containing 100 to 13 per cent. Sb are composed of crystals of Sb in an increasing amount of the eutectic or ground mass, a type of structure which is shown in photographs 1, 7 and 8. At 13 per cent. Sb the alloy is all eutectic and may be represented by photograph 3 or 5 (magnified 60 and 260 diameters respectively). Alloys from 0 to 13 per cent. antimony are composed of crystals of lead in dendritic form set in an increasing amount of the eutectic. Photo 6 represents this type.

In short, above the curve *ArB* the alloys are completely liquid, below the horizontal line at 247° C. everything is solid. In the left triangle *Ar*–247° C., we have crystals of Sb in a liquid alloy: in the right triangle crystals of lead in a liquid.

Lead antimony alloys are used as bearing metal, type metal, hard lead, etc.

Lead-Tin.

The curve for lead and tin is similar to that of lead and antimony. As determined by Kupffer* the eutectic contains 37 per cent. lead and freezes at 182° C. Charpy† gives 38 per cent. lead, 62 per cent. tin. Roberts-Austen‡ gave the point at 68 per cent. tin and 180° C. and published curves showing tensile strength and extensibility. Tucker and Rosenhain§ found the exact point to be 63 per cent. tin, freezing at 182.5° C.

Thus alloys with less than 63 per cent. tin are composed of crystals of lead set in the eutectic: those with more than 63 per cent. tin are composed of crystals of tin set in the eutectic. Solid lead holds some tin in solution, the amount varying with the rate of freezing. Rosenhain and Tucker were able to get 16 per cent. in solution by long annealing. As ordinarily made, however, the amount is very much less.

The lead-tin alloys are used as solders, pewter, in the manufacture of toys, etc.

Tin-Antimony.

From a microscopic study of alloys of antimony and tin, Charpy‖ found a definite compound containing about 50 per cent.

* *Annales de Chim. et Phys.*, XL., 289.

† *Metallographist*, II., 41.

‡ Fourth Report, Alloys Research Committee, February, 1897.

§ *Phil. Trans. Royal Soc.*, 209 A (1909), 89.

‖ *Metallographist*, I., 197.

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Sb, which forms isomorphous mixtures with Sb. He found (incorrectly) a eutectic at about 10 per cent. Sb.

Stead* showed that up to 7.5 per cent. Sb the alloys consist of a solid solution, above this point cubes of a compound SbSn separate out when the alloy freezes. At the antimony end of the series crystals of Sb are the first to form.

The constitution of these alloys was not understood till Reinders† published his thermal diagram. Later Gallagher‡ modified the curve and showed a thermal transformation in the cubes (SbSn of Stead) at about 315° C.

Williams§ gave a simpler curve and omitted the transformation at 315° C. The main points of these three curves are the same however, and Fig. 2 is taken from them, somewhat simplified, to explain the main changes.

The addition of tin to antimony lowers the freezing-point progressively. The curve shows two inflections at *f* and *d* with two horizontals *fgh* and *deg'*. There is no true eutectic as pure tin freezes lower than any of the alloys of the series. Such a curve shows that above *AfdC* all is liquid: below *Ahgg'eC* all is solid. Between these two curves we have a mixture of solid and liquid. It also shows that at 422° C. the alloys from 90 to 50 per cent. Sb show a reaction whereby the solid corresponding to *h* reacts with the liquid corresponding to *f* to form solid *g*. Now *h* is antimony containing about 10 per cent. Sn in solid solution, Sbβ for short: *g* corresponds to SbSn. The liquid *f* contains about 50 per cent. Sb. Similarly at 244° C. we have the reaction of *g'* or SbSn with the liquid *d* (8 per cent. Sb) to form the solid *e* containing 10 per cent. (a solid solution, tin with 10 per cent. of antimony or Snα for short). Then from 100 to 90 per cent. Sb we have Sbβ. From 90 to 52 per cent. Sb we have Sbβ surrounded by a ground mass of SbSn. From 52 to 10 per cent. Sb we have SbSn in a ground mass of Snα, whilst from 10 to 0 per cent. Sb we have Snα, tin containing antimony in solid solution.‖

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* *Journal Soc. Chem. Ind.*, December 31, 1898; *Metallographist*, II., 314.

† *Zeits. anorg. Chem.*, 1900 (25), 113.

‡ *Journal Physical Chemistry*, 1906, X., 93.

§ *Zeits. anorg. Chem.*, 1907, 35, 12.

‖ Type photographs are given by W. Campbell, *Journal Am. Chem. Soc.*, 1904, XXVI., p. 1308.

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For the sake of simplicity we have omitted the change in the SbSn at 315° C. and also the fact that SbSn probably is capable of holding some 2 per cent. of tin in solid solution. In addition whilst Sb dissolves 10 per cent. tin at 422° C., the amount at ordinary temperatures is much less.

Tin antimony alloys have been used as the basis of Brittania metal, for antifriction alloys, for processes of engraving, etc.

Ternary Alloys.

The simplest method of representing a series of ternary alloys is by means of an equilateral triangle. In Fig. 3 the corners of the triangle represent the pure metals *A*, *B* and *C*. The binary alloys of *A* and *B* are located in the line *AB* which is a projection of Fig. 1. Similarly the binary alloys of *B* and *C* are represented by the line *BC* and the alloys of *A* and *C* by the line *AC* which is a projection of Fig. 2. An alloy of all three metals is represented by a point within the triangle, for the sum of the perpendicular distance from any point within the triangle to the three sides is constant and is taken as equal to 100. Take the alloy represented by *O* in Fig. 3, the percentage of *A* is given by the perpendicular from *O* to *BC*, the percentage of *C* by the perpendicular from *O* to *AB*, while the percentage of *B* is given by perpendicular from *O* to *AC*. Hence if we wish to indicate temperature the scale must be perpendicular to the plane of the triangle, in other words we must construct a solid figure.

The simplest case is where each pair of metals have a curve similar to Fig. 1. That is to say the pairs are completely soluble in the liquid state and insoluble in the solid. Charpy* has worked out the PbSnBi diagram and Stoffel† those for PbSnCd and BiSnCd.

More complicated cases have been worked out by Geer‡ and by Sahmen and Vegesack.§ The latter show the case where the pairs *AB* and *BC* form eutectics and have curves represented by Fig. 1, while the pair *AC* show no eutectic but a reaction between the

* *Metallographist*, II., 28.
† *Zeits. anorg. Chem.*, 53, 137.
‡ *Journal Physical Chem.*, VIII., 237.
§ *Zeits. physikal. Chem.*, 1907, 59, 257.

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solid *h* and the liquid *f* to form the solid *g* (Fig. 2). Then at *O* (in Fig. 3) there is a similar reaction whereby solid *h'* and liquid *O* react with the formation of solid *g'* and *B*.

In the case of lead, tin and antimony, we have two pairs represented by Fig. 1 and the third by Fig. 2, where we have two reactions. Fig. 3 gives such a ternary diagram and the following is a brief description of the changes that take place when an alloy solidifies. *A* represents Sb, *B* is Pb and *C* is Sn. Then *r* is the eutectic of Pb and Sb with 13 per cent. Sb freezing at 247° C. Again *q* is the lead-tin eutectic at 63 per cent. tin, freezing at 182° C. On the side *AC* the point *g* represents SbSn, *f* the reaction point at 422° (*f* in Fig. 2), *d* the reaction point at 244° (*d* in Fig. 2), and *e* is 10 per cent. Sb, or saturated Snα.

In the area *ArOf* the first solid to form is antimony (with a little tin in solid solution). In the area *BrOPq*, lead crystallizes out first. In *fOPd* SbSn forms first; while in *CdPq*, tin crystallizes out first with more or less Sb in solid solution (Snα).

Assuming for the sake of simplicity that pure Sb crystallizes out, in the triangle *AOr* an alloy will deposit crystals of Sb on freezing until the composition of the liquid reaches a point on the line *rO*. Then we shall have Pb and Sb freezing out side by side and the liquid composition sliding down* the line *rO* till it reaches the reaction point *O*. At this point we have the reaction Sb crystals + liquid *O* forming solid SbSn + Pb. If the liquid is used up before the Sb the alloy goes solid at *O*. If not, when the Sb has disappeared the liquid follows the line *OP* and we have SbSn and Pb separating out side by side till the alloy finally is all solid.

In the area *BOr* the first to freeze are crystals of lead and the liquid reaches the line *rO*, the further changes being as above.

In the area *AOf*, antimony crystals freeze out and the liquid reaches the line *fO* and follows down it towards *O*. This change in composition of the liquid is brought about through a reaction between the solid Sb and the liquid to form SbSn. On reaching *O* as before we have the reaction above mentioned, but if the Sb is all gone before the liquid reaches *O*, further solidification occurs as in the next case.

In the area *fPO*, SbSn crystallizes and the liquid reaches the

* The arrows in Fig. 3 show fall of temperature.

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line *OP* and follows it, due to the simultaneous freezing of SbSn and Pb. On reaching the point *P* we have another reaction similar to the one at *O*. Here SbSn + liquid *P* react to form solid Snα + Pb. As before if all of the SbSn is used up before the liquid *P* further freezing follows the line *Pq* and the alloy finally goes solid.

In the area *BOP*, lead freezes out first and the liquid on reaching line *OP* follows it as above.

In the area *BPq*, lead crystallizes out first* and on the liquid reaching the line *Pq* lead and tin (Snα) freeze out side by side till *q* is reached, the binary eutectic of Pb and Sn, also the eutectic of the system.

In the area *CdPq* we have the tin solid solution (Snα) first freezing out and the liquid reaches the line *Pq* as before.

In short, the line *rO* the boundary between the Sb area and the Pb area represents the simultaneous freezing of Pb and Sb. The line *OP* is the boundary between the Pb and the SbSn areas and represents the simultaneous freezing of Pb and SbSn; while *Pq* to the boundary between Pb and Snα and is the simultaneous freezing of these two. All of these give structures resembling that of a true eutectic.

On the other hand the boundary line *fO* represents the reaction whereby SbSn forms at the expense of the Sb and *dP* represents the reaction whereby Snα forms at the expense of SbSn.

The work then was to determine the points *O* and *P* and approximate the lines *rO*, *fO*, *OP*, *dP* and *Pq*. At the outset it was intended to determine the constituents and boundary lines in the lead corner of the triangle with not more than 25 per cent. Sb and 25 per cent. Sn and, with that in view, a number of cooling curves were run with a plate recorder. These were fairly satisfactory and with the aid of the microstructure the point *O* was determined, but this left the work so incomplete that further curves were run and alloys examined to determine the approximate directions of the lines and to locate the point *P*.

Due to the fact that stirring was not resorted to, surfusion was liable to occur and many of the curves were inexact and required

* As before for the sake of simplicity we assume the lead to be pure and not contain Sn in solid solution.

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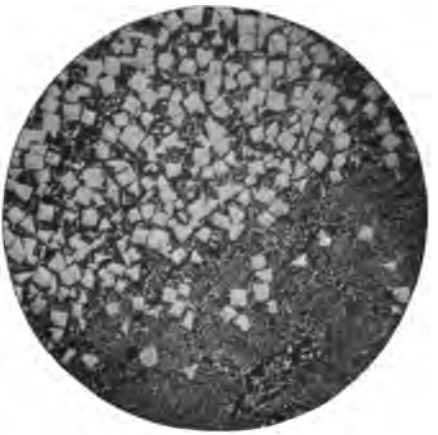

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the microstructure to clear them up. However enough data were gathered in the work on about 100 alloys to locate the two points.

(1) O at 80 per cent. Pb, 10 per cent. Sn, 10 per cent. Sb, at 245° C.

(2) P at 40 per cent. Pb, 57.5 per cent. Sn, 2.5 per cent. Sb, at 189° C.

Using fixed points (1) PbSn eutectic at 182° C., (2) freezing point of tin 232° C., (3) PbSb eutectic at 247° C., (4) freezing point of lead 327° C., (5) zinc at 419° C., (6) antimony at 631° C. for calibration.

Microstructure.

In the preparation of sections for the microscope the pieces must be sawed through vertically because there is often a marked difference between the top and the bottom of the specimen. This is due to the fact that both the antimony and the SbSn crystals are much lighter than the liquid out of which they freeze and they therefore tend to float to the surface of the melt. After cutting with a saw the surface of the section must be carefully filed to take off all of the surface altered in the process of cutting. If this is not done the structure will be broken up and obscure. After polishing in the usual way the final polish on a rouge board must be continued longer than usual till a good surface is obtained. As an etching reagent two per cent. nitric acid in alcohol is very good.

With small melts cooled in the air the structure shows that equilibrium is not always established. Alloys in that part of the area *ArOf* whose Sb ought all to have disappeared due to the reaction on *fO* or at *O* often show undissolved cores of Sb. Similarly with SbSn in the area *fOPd*. With very slow cooling of course conditions would have brought about more complete equilibrium.

The microstructure of the series is typically shown in the following photographs.

No. 1. Pb 65, Sn 10, Sb 25 × 60 diameters. The white crystals are Sb set in a ground mass which freezes at the reaction point *O*.

No. 2. Pb 78, Sn 11, Sb 11 × 60. Occurs very near *O*, practically on the line *fO*. This view is the top of the alloy and

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shows the layer of SbSn cubes which have floated to the surface.

No. 3. Is the structure of the rest of the alloy which froze on the line *OP*.

No. 4. Shows the same, × 260 diameters. The ground mass is seen to be a mechanical mixture of Pb and SbSn with a structure like that of a true eutectic.

No. 5. Pb 80, Sn 10, Sb. 10. The alloy *O*, × 260.

No. 6. Pb 85, Sn 10, Sb 5 × 260, shows excess of lead as black dendrites, surrounded by the same ground mass as before.

No. 7. Pb 75, Sn 10, Sb 15 × 60 shows the star-like forms of the antimony crystals and the skeleton crystals of composite nature which form at *O*.

No. 8. The same. Around the cubes, etc., of antimony, we have the eutectic-like structure of Pb and SbSn like an envelope. Such an envelope would make equilibrium extremely improbable with even moderately slow cooling for the reaction of the solid Sb and the liquid would have to take place through this envelope by diffusion.

The structure of the alloys formed at the reaction point *P* are similar to those at *O* except that we find coarse and fine laminæ of Pb and Snα which do not show the marked contrast seen in photo No. 5. Alloys in the *CdPq* area resemble photo No. 6, the dendrites being Snα and the ground mass that which forms on *Pq*. The alloys in *fPd* show cubes of SbSn surrounded by Snα, which formed on *dP*, and a ground mass due to *P*. Similarly the alloys in the other sections.

Conclusions.

The ternary diagram can be divided up into four main areas depending on the constituent which first crystallizes out. These areas are bounded by the side of the equilateral triangle representing the binary alloys and by the lines (1) from the eutectic of Pb and Sb and from the reaction point Sb-SbSn intersecting in the upper ternary reaction point *O* at 80 Pb, 10 Sn, 10 Sb; (2) from the eutectic of Pb and Sn and the reaction point SbSn-Snα intersecting in the lower ternary reaction point *P* at 40 Pb, 57½ Sn,

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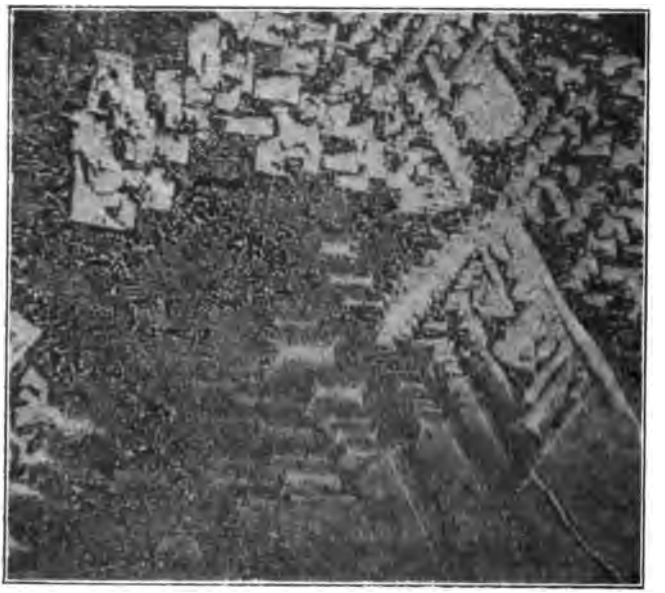

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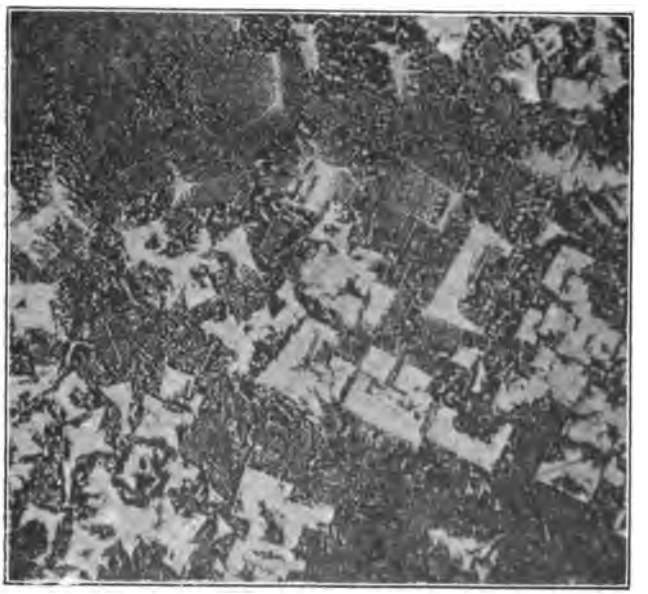

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2½ Sb; (3) from the upper to the lower reaction points, viz., *O* to *P*.

The upper reaction point *O*, Pb 80, Sn 10, Sb 10 occurs at 245° C., some 2° C. below the freezing point of the eutectic of lead and antimony. Hence the slope *rO* is very slight.

The lower reaction point occurs about 189° C., some 7° C. above the binary eutectic of lead and tin.

The eutectic of the whole series, *i. e.*, the alloy with the lowest freezing point is the binary eutectic of lead and tin freezing at 182° C.

In the diagram the lines *fO* and *dP* have been drawn straight. It may prove that, when they have been more accurately determined by the examination of numerous alloys in their neighborhood, they curve more or less. However the general form of the diagram is correct.* In the discussion of the ternary diagram many of the minor points have been omitted for the sake of brevity and clearness.

From the structures it is seen that the alloys follow four main types—(1) those with excess of hard Sb or SbSn set in a laminated ground mass, (2) those with excess of soft lead dendrites in a laminated ground mass, (3) those with the excess of Snα in a laminated ground mass, (4) those with excess of hard SbSn in Snα and ground mass; bearing in mind that there are two types of ground mass. This accounts somewhat for the wide variation in composition for certain uses as shown by the alloys in the table.

* The practical work on which this paper is based was done and submitted as a thesis in the Department of Metallurgy, Columbia University, for the session 1909–1910.

R. Loebe in *Metallurgie*, VIII., 7 (January, 1911), has published results of an elaborate and careful research in the Pb-Sb-Sn alloys, locating the boundary lines exactly and discussing the subject of ternary alloys very thoroughly. Our results agree in regard to the point *O* but the composition of the point *P* occurs near the tin side of the diagram. On account of greater refinement of methods Mr. Loebe's determination is no doubt nearer the truth.

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EDITORIAL.

It is necessary to call attention of members of the Alumni of the Schools of Mines, Engineering and Chemistry to the trouble caused by inaccurate addresses. Every issue of the QUARTERLY brings back a number of cards from the various postmasters stating that the addressee is unknown and the QUARTERLY is held for return postage.

Changes of addresses should be sent on a post card to the QUARTERLY or to Professor Mayer, Secretary of the Alumni Association.

It has been suggested repeatedly that we have a Personal Column in which shall be recorded notes of professional interest about our various members. Of course the lists of alumni partly cover this ground but we ought to supplement these during the interim of publication.

Therefore if you agree with this idea send in a post card stating any important change in your position since the last alumni list came out. If you have made no change perhaps you know some item of interest in regard to some member of your class. We think the idea worthy of a trial.

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ABSTRACTS OF RECENT MINING DECISIONS.*

LOCATION OF CLAIMS.

Location of Mining Claim—Effect of Excess.

A placer mining claim located in good faith is not wholly void merely because it exceeds twenty acres. In such case it is void only as to the exces, and this may be rejected from any provision the owner may select, and until he has been advised of the excess and has had a reasonable time to make his selection, his possession extends to the entire claim, and a third person who makes a location of any part of such claim is a trespasser and his location is a nullity and is void for any purpose.

Jones *v.* Wild Goose Mining & Trading Co., 177 Federal, 95. March, 1910.

Location of Mine on Public Mineral Lands—Marking Boundaries.

The United States statutes provide that locators of all mines on the public domain so long as they comply with the laws of the United States and with the state and local regulations not in conflict therewith, shall have the exclusive right of possession, etc. They also provide that miners of each mining district may make regularities not in conflict with the laws of the United States or with the laws of the state. Such statutes grant authority, by inference, to the state to add to the general location requirements when not in conflict with United States laws. Accordingly where the United States statutes require that, when a discovery has been made within the limits of the claim sought to be taken, the location be distinctly marked on the ground so that the boundaries may readily be traced and the record of the location shall contain the names of the locators together with a description of the claim by reference to some natural or prominent monument as will identify the claim, the requirements of the Colorado statute providing that before the filing of a location certificate the discoverer shall locate his claim by posting a notice thereon and marking the surface boundaries with substantial posts at each angle of the claim, are valid and are not in conflict with the federal statutes but merely add to its general terms.

Saxton *v.* Perry, (Colorado) 107 Pacific, 281. February, 1910.

Statute Governing Location and Relocation of Mining Claim.

Section 1 of the statute of Washington prescribes the duties of the discoverer of a lode in the location of a mining claim; section 2 of

* Prepared for the School of Mines Quarterly, by J. W. Thompson, Attorney-at-Law, Indianapolis, Indiana.

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the statute requires the discoverer, upon filing notice of his discovery, to locate his claim by sinking a discovery shaft upon the lode to the depth to at least 10 feet; section 8 provides that the relocation of forfeited or abandoned quartz or lode claims shall only be made by sinking a new discovery shaft and fixing new boundaries in the same manner and the same extent as is required in making a new location, or by sinking an original discovery shaft 10 feet deeper than it was at the date of commencement of such relocation; section 9 provides that the provisions of the statute relating to the discovery shafts shall not apply to any mining location west of the summit of the Cascade Mountains. In a contested location this statute was construed by the Supreme Court of Washington to mean that section 9 applied to relocation of forfeited or abandoned quartz claims under section 8 as well as to original locations, so as to excuse the sinking of a discovery shaft upon relocation of a forfeited or abandoned claim west of the summit of the Cascade Mountains.

National Milling & Mining Co. *v.* Piccolo, (Washington) 107 Pacific, 353. March, 1910.

Protection by Relocation.

Where a mining claim has been improved by the original locators or their assigns without abandonment, but because of a defect in the original location certificate, the former locators or their assigns have the right to protect their interest by making a new location, if there has been no failure on their part to perform the required development work. So where a corporation acquired all the title of the original locators of a mining claim both under an original and subsequent location it was competent and proper for such corporation to record an additional or amended certificate of location and claim thereby the discovery as of the date of the discovery by the original locators. In such case it is proper to prove that the original locators in relocating a claim adopted the boundary stakes of the original location.

Bergquist *v.* W. Virginia-Wyoming Copper Co., (Wyoming) 106 Pacific 673. February, 1910.

Cutting Timber from Mineral Lands.

The authority given by the United States statutes to cut timber upon mineral lands not subject to entry except for minerals does not extend to land adjacent to lands valuable for mineral purposes but only includes lands known to be themselves valuable for minerals, which are the only lands excluded by the United States statutes from any but mineral entry.

United States *v.* Plowman, 30 Supreme Court Reporter, 299. February, 1910.

CONFLICTING OF RIGHTS.

Taking Ore Without Consent—Measure of Damages.

The measure of damages for unlawfully taking ore from a mine, but which taking is done in good faith under the belief that such ore

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belonged to the person taking the same, is the value of such ore as it was in the mine before mined or undisturbed. As applied to the taking of oil the measure of damages for the taking of oil from land through mistake as to its ownership is the value of such oil at the surface or in tank less the cost of pumping and taking it, and not the usual royalties for producing oil.

Bender *v.* Brooks, (Texas) 127 Southwestern, 168. April, 1910.

MINING LEASES.

What Constitute a Mining Lease.

A lease of a tract of land for a term of ten years by which the lessee agreed to mine the same and sell the minerals produced, and pay the lessor a percentage of the proceeds as royalty was a mining lease which gave the lessee no property rights in the ore under the land excepting such as it should mine and sell within the terms of the lease, and such lessee could not maintain an action against a trespasser who mined and removed ore from the premises during the stated term for the recovery of the value of the ore taken, but was entitled to recover damages only to the extent of the actual injury.

Providence Mining & Milling Co. *v.* Nicholson, 176 Federal, 29. February, 1910.

Construction of Gas and Oil Lease.

A gas and oil lease provided for the payment of $100 yearly for each and every well drilled and from which gas was transported or used off of the land and that the lessee should deliver to the lessor one sixth of all the oil produced and saved from the premises. The lessee entered upon the premises and drilled six wells which produced both gas and oil. In an action on the lease and contract to recover the rentals it was held that the lessee could not fraudulently refuse to transport the gas from the premises for the sole purpose of defrauding the lessor out of the rentals, and that the payment of the royalty for the oil did not prevent a recovery of the royalty for the gas.

Pittsburg-Columbia Oil & Gas Co. *v.* Broyles, (Indiana) 91 Northeastern, 574. May, 1910.

ASSESSMENT WORK.

Action to Determine Right to Public Mining Lands—Improvement Work.

In an action to quiet title to mining claims an instruction was correct which informed the jury that if the requisite work was done upon one claim, the title to which was not in issue, and such work was done in pursuance of a system that tended to a development of the claims owned or claimed by the plaintiff in action, such work was sufficient to prevent a forfeiture of any such claims, where it was shown that the claim on which the work was done was adjoining claim, and the law is that work done upon any one of such claims can be applied to any of the others. An instruction was correct which

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also informed the jury that work outside the limits of a claim must be such as has a tendency to benefit or develop the claim and that the burden of proof was upon the defendant to show that the necessary work was not done as required by law, but that the burden was upon the plaintiff to show that any work done outside of the claim must be for the benefit or for the development of the claim. In such an action the law requires clear, convincing evidence to affect the forfeiture of a claim duly located and worked in good faith and the evidence must satisfy a jury by a clear preponderance thereto that the plaintiff failed to perform the necessary work before a forfeiture will be permitted. So whether work done on a mining claim is such as to benefit or develop other claims and so satisfy the requirement of the law as to development of such claims is a question of fact.

Big Three Mining & Milling Co. *v.* Hamilton, (California) 107 Pacific, 301. January, 1910.

LIABILITY OF INJURY TO MINERS.

Injury to Miner—Assumption of Risk.

In the operation of a mine it was the duty of the mine owner to furnish props to support the roof and it was the duty of the miner to set such props as the work progressed and the end of the room of the mine was extended. In the operation of the mine the miner was killed by the falling of the roof. The undisputed evidence showed that the intestate was an experienced miner and knew of the necessity of propping the roof as he advanced further in his work and of the danger of its falling if not properly supported, and that when he went to work the morning of his death he, with his assistant, examined the roof and it seemed alright. It appeared also that he worked from two to four days without putting up props and then called on the mine foreman for props and was told to go ahead with the work and the props would be sent and that the deceased did thereafter work extending his room when the unsupported roof fell and killed him. In continuing his mine and extending his room without propping the intestate well knew and appreciated the danger and assumed the risk although the mine owner neglected to furnish the necessary props. It is not a case of a master furnishing a defective appliance or a place which he promises should be repaired and made safe; but it is a case where assurance of safety was given to the servant who was making his own place to work which he knew as well as any one could know would be and was dangerous without using the appliances the master promised to furnish, and which he knew had not been furnished, and he well knew in continuing to work therein he was in danger and was increasing the danger with every stroke of the pick, as he was an experienced miner and well knew the necessity of propping the roof as he advanced. Under such circumstances the intestate had no right to rely either on the statement of the foreman that the top was alright or that the props would be furnished at some future time. Under such circumstances the federal court held that as a matter of law the intestate, in continuing to work in the mine in the absence of suffi-

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cient proping to support the roof in his room assumed the risk of injury from the falling roof and the mine owner was not liable.

Republic Iron & Trust Co. *v.* Thomasino, 176 Federal, 49. February, 1910.

Miners Assumption of Risk.

A driver in a coal mine does not assume the risk of injury due to the failure of a mine boss or mine operator to perform the statutory duty of making a dangerous place safe where the statute clothes the mine boss with authority to exclude miners from any dangerous place in the mine until it is made safe; and such a driver in a coal mine does not assume the risk of injury, where the mine boss directs him to enter the place after it has been made safe, but by which he was in fact injured. The doctrine of assumed risk does not apply merely because the manner of making the place safe is left to the discretion of the mine owner and the judgment of the mine boss.

Princeton Coal Mining Co. *v.* Howell, (Indiana) 92 Northeastern, 122. June, 1910.

Injury to Miner—Contributory Negligence.

A miner is not guilty of contributory negligence because of his failure to give notice within a reasonable time of a known defective condition. And such miner working in a mine may assume that the mine owner has discharged his duty in seeing to it that the place is reasonably safe, and the miner is not required to exercise even reasonable care to ascertain whether the roof of the mine is safe or not, except as he is required by statute to examine the working place before commencing work and to repeat the examination after a stoppage of work during the shift.

Lockhart *v.* Sloss-Sheffield Steel & Iron Co., (Alabama) 51 Southern, 627. March, 1910.

Miners as Fellow Servants.

Miners engaged in mining operations are fellow servants; and the rule applies although by reason of seniority one servant was director or boss of their operations, and the negligence of such director or boss in failing to get a new rope, or for permitting an old rope to be used by his fellow miners for an improper purpose and for a use to which it had not been put, and the master was not liable even where such boss or director was guilty of negligence.

Avikanen *v.* Baltic Mining Co., (Michigan) 125 Northwestern, 545. March, 1910.

COAL LANDS.

Mineral Rights in Railroad Right of Way.

A deed conveying to a railroad company a right of way containing the clause "coal rights reserved," was held to reserve the title to the coal under the right of way and that it separated the title to the coal from the title to the surface, together with the mining rights connected with it, and that the grantor could mine and remove the coal

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and could use the space where the coal was found in any way he pleased and as though no conveyance of the surface had been made.

Atterbury *v.* Blair, (Illinois) 91 Northeastern, 475. April, 1910.

Statute Regulating Sprinkling and Ventilating Mines.

The statute of Indiana (Burns' Ann. St. 1908, section 8579) provides that road-ways or entries in any mine shall be regularly and thoroughly sprinkled when they are so dry that the air becomes charged with dust, and it is made the duty of the mine inspector to see that the provision is complied with. The fact that this provision is a part of the same section in which provision is made for ventilating the mine in no wise abrogates the duty to sprinkle, irrespective of the subject of ventilation, and was imposed for the benefit and protection of persons employed in the mine. A failure of the mine operator to so sprinkle the road-way and entries when so filled with dust, and from which a dust explosion resulted, injuring a miner, the failure to sprinkle as required by the statutes was negligence on the part of the owner or operator and entitled the injured miner to recovery. The failure to sprinkle may be charged as the proximate and direct cause of the explosion and the injury of the miner. The dust in such case would be regarded as the efficient or proximate cause of the explosion and injury.

Vandalia Coal Co. *v.* Yemm, (Indiana) 92 Northeastern, 49. June, 1910.

MISCELLANEOUS.

Mining Monazite—Water Privilege.

Monazite is found in gravel in beds of streams and adjoining the bottom lands under the top soil. In mining it the top soil must be removed and the gravel thus exposed is shovelled on a perforated steel plate and water is then turned upon it so as to work the smaller material into the trough and the coarser gravel and rock shovelled out and thrown aside. The highest percentage of monazite is obtained by washing several times. The right to mine monazite under a lease of minerals and minerals rights for a fixed period at a stated royalty necessarily involves and carries with it such water rights as are reasonably necessary for the proper conduct of such mining operations, and the performance of such an agreement will be specifically enforced by a court.

National Light & Thorium Co. *v.* Alexander, (South Carolina) 67 Southeastern, 310. March, 1910.

Title to Oil.

The owner of land has no specific title to the oil therein until it has been removed from the earth. Petroleum oil is a mineral, and while in the earth it is part of the realty. If it moves from place to place by percolation or otherwise it forms a part of that tract of land in which it remains for the time being, and if it moves to another tract it becomes a part and parcel of such tract, and it forms a part

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of some tract until it reaches a well and is raised to the surface, and then it first becomes subject to distinct ownership separate from the land and is personal property, the property of the person who has taken it from the well into which it came. This rule applies whether the oil moves, percolates, or exists in pools or deposits. In either event it is the property of, and belongs to the person who reaches it by means of a well, and severs it from the realty and converts it into personalty.

Bender *v.* Brooks, (Texas) 127 Southwestern, 168. April, 1910.

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ABSTRACTS—ANALYTICAL CHEMISTRY.

By E. WALLER.

Colloid Precipitates. Liebschutz (*Ch. News,* CII., 213). Metastannic acid as obtained frequently in analysis of alloys and other materials containing Sn, often takes a colloidal condition, and gives trouble by passing through filters. The difficulty was remedied by adding a very little albumen solution, and heating for a short time.

Similarly when separating Zn from Cu by the cyanide method, the ZnS becomes gelatinous and very difficult to wash. On adding a few drops of $Pb(C_2H_3O_2)_2$, the PbS helps to coagulate the ZnS, and filtration and washing are more easily accomplished.

Waste Platinum Recovery. Blair (*J. Ind. and Eng. Chem.,* II., 102). In the alcoholic washings of K_2PtCl_6 the Pt may be completely separated as metal by standing the solution for some weeks in direct sunlight. The same reduction will take place in the water solutions of K_2PtCl_6 if a little alcohol is added, and the "sunlight treatment" is given.

Cleansing Platinum Wire for Flame Tests. de Koninck (*Bull. Soc. Belg. Chim.,* XXIV., 197). Many substances as Cu, Ba, etc., adhere pertinaciously. It is recommended to fuse a borax bead on the wire and run it along while liquid, then shake it off and finally volatilize off the borax remaining by long heating.

Methylene Blue as Indicator. Sinnatt (*Analyst,* XXXV., 309). $TiCl_3$ and other reducing agents decolorize methylene blue, and it has been used in that relation frequently. The author finds that I in KI solution alters the color of methylene blue to yellowish green, and finally to yellowish brown, and proposes the blue as a substitute for starch in iodometric titrations. In strong solutions a precipitate of an iodo-compound of the methylene blue is obtained, but in the dilute form as an indicator the color changes are as described.

Indicator for Free Acid in Superphosphates. Guthrie & Ramsay (*J. Proc. Roy. Soc. N. S. Wales,* XLIII., 69). The sharpest indicator is sodium alizarin sulfonate. This indicator is prepared by treating alizarin with fuming H_2SO_4, and then converting to the Na salt by neutralizing with Na_2CO_3. The indicator is almost neutral to the acid Na and Ca salts.

Potash in Soils. Bieler Chatelan (*C. Rend.,* CL., 716). The *assimilable* K_2O in soils is best determined by shaking with CO_2 water and determining the amount extracted. An amount of the fine soil corresponding to 30 gms. of dried soil is shaken constantly for 10 hours with 500 c.c. of CO_2 water. This is then filtered, and 400 c.c. of the filtrate evaporated, SiO_2 separated, etc., the K being eventually determined. Soils affording less than 0.15 to 0.20 part K_2O per 1,000 will prove more fertile if treated with potash salts in conjunction with

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phosphatic manures. In a later communication (*C. Rend.*, CL., 716) the author states that extraction by *displacement* has been found to give more accurate results.

Alkalinity of Sea Water. Ruppin (*Zts. Anorg. Chem.*, LXVI., 122). Place 200 c.c. in a Jena flask with ground in stopper, add 15 c.c. *N*/20 HCl and boil to expel CO_2. Cool, add 15 c.c. *N*/20 KIO_3 and 10 c.c. of 10 per cent. KI. Stopper and stand aside in the dark for 75 minutes. Then titrate with *N*/50 $Na_2S_2O_3$ freshly prepared by diluting *N*/5 solution. Tables of the results in the North Sea and Baltic are given.

Alkalies. Cavazza (*Chem. Centr.*, I., 962). The author finds that alkaline chlorides when repeatedly evaporated with oxalic acid become converted to oxalates which can be ignited to carbonates, and can then be titrated alkalimetrically. If the weight of the pure carbonates is determined, that figure, combined with the determination of alkalinity, will give the necessary data for calculating the respective proportions of Na_2O and K_2O present.

Lime Analysis. Heyer (*Chem. Ztg.*, XXXIII., 1157). $CaCO_3$ was found to be appreciably soluble in neutral, cold, saturated solutions of NH_4 salts. The solubility is almost completely annuled by free NH_3. A good solution for separating small amounts of CaO (SrO, etc.) is an NH_4Cl solution containing at least 1 gm. NH_3 per liter to which has been added 40 c.c. NH_4OH (Gr. 0.910).

"Available Lime" Cyanide Work. Barney (*J. Ind. & Eng. Chem.*, II., 407). In commercial limes, CaO can be titrated in presence of $CaCO_3$ by oxalic with phenolphthalein indicator. Prepare a solution containing 14.6068 $H_2C_2O_4.2H_2O$ per liter. On using 0.650 gm. of the sample the burette will then read in percentages.

Grind the sample to 200 mesh, place 0.65 gm. in an Erlenmeyer, add 50 c.c. of water, stopper and shake vigorously for 10 seconds, add 2 drops phenolphthalein solution (0.5 gm. in 100 c.c. 50 per cent. alcohol) and run in the standard oxalic solution until the color is discharged. Shake well, and if the color returns, add more oxalic. Unneutralized CaO gives a strong vivid color, whereas that given by MgO is pale and weak, requiring some time to develop.

Lime in Magnesites, etc. Murmann (*Zts. Anal. Chem.*, XLIX., 688). In presence of large amounts of MgO, the usual procedure of separation utterly fails. The assertion is made that where 1.5 per cent. CaO is really present, results ranging from 0.1 to 4 per cent. may be obtained.

The method described requires a knowledge of approximately the amount of CaO present. To the solution of the chlorides add *N*/10 H_2SO_4 in amount more than sufficient to combine with all the CaO. Evaporate to dryness, and extract repeatedly with 90 per cent. alcohol. Take up the residue with water, and precipitate out the CaO by $H_2C_2O_4$, adding NH_4OH cautiously until the reaction of the solution is slightly alkaline.

Alkaline Earths. Lindet (*Bull. Soc. Chim.*, VII., 434). A table of the solubilities of the oxides in phenol and resorcinol is given. That for phenol is as follows:

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Strength of Sol'n.	CaO.	BaO.	SrO.	MgO.
20 per cent.	5.32 per cent.	forms a mass.	4.51 per cent.	0.60 per cent.
15 per cent.	4.68 per cent.	11.67 per cent.	2.31 per cent.	0.58 per cent.
10 per cent.	3.27 per cent.	8.28 per cent.	1.32 per cent.	0.56 per cent.
5 per cent.	1.52 per cent.	4.42 per cent.	0.55 per cent.	0.33 per cent.

Resorcinol and hydroquinone dissolve larger amounts, but the phenol is more readily available, and can be used to advantage in many analytical processes, such as analysis of commercial limes, etc. Usually the 5 per cent. solution will be most convenient. It has practically no solvent action on carbonates, phosphates, silicates or Fe_2O_3 or Al_2O_3.

Ether Separation of $FeCl_3$. Wysor (*J. Ind. and Eng. Chem.*, II., 45). In separations of Fe from Ni and other metals based on the solubility of $FeCl_3$ in ether, the author finds that the assumption that P_2O_5 does not dissolve in the ether together with the $FeCl_3$ is erroneous, as some (apparently proportional to the amount of Fe) always dissolves.

Chromium in Iron and Steel. Feschbach (*Stahl u. Eisen*, XXIX.. 248). For small amounts dissolve 5 to 10 gms. of the sample in HCl, filter—add metallic Zn to reduce what Fe may have oxidized, and then an excess of ZnO to precipitate. Boil well, settle, filter, wash with hot water, dry and ignite. Then mix the mass with 2 parts Na_2CO_3 and 3 parts MgO, and fuse for an hour in a muffle or with plentiful access of air. Leach out Na_2CrO_4 with water and determine Cr by acidifying, adding KI and titrating with $Na_2S_2O_3$.

Titanium in Rutile, etc. Barnebey and Isham (*J. Am. Chem. Soc.*, XXXII., 957). Weigh out in Pt crucible. Moisten with 5 or 6 drops conc. H_2SO_4, add 1 c.c. HF. Heat to cessation of SO_3 fumes. Then add 5 to 10 gms. dry Na_2CO_3 and a little $NaNO_3$ and fuse for 30 minutes. Leach out with hot water. Filter and wash with hot water. The insoluble portion contains Fe_2O_3 and Na_2TiO_3. Dissolve in HCl (Gr. 1.11), cleansing the crucible also with HCl. Evaporate the solution to 15 or 20 c.c. Add 2 c.c. conc. HCl, transfer to a separator funnel, and shake with ether. Run off and shake again with fresh ether, and repeat until the ether shows no color. Test the aqueous solution with H_2O_2. If much TiO_2 is present add H_2SO_4, evaporate to fumes, dilute, add SO_2, nearly neutralize with NH_4OH, add 1 or 2 gms. $(NH_4)HSO_3$, warm for 30 minutes, then add 10 or 15 gms. $NH_4C_2H_3O_2$ and 5 to 10 c.c. glacial acetic, and boil for 15 minutes, when the TiO_2 will be all precipitated. Wash with dilute acetic. If the TiO_2 is small in amount it may be determined by colorimetric comparison.

Titanium. Gemmell (*Analyst*, XXXV., 198). The H_2SO_4 solution, which should not contain over 0.15 gm. TiO_2 is reduced in an Erlenmeyer flask fitted with a Bunsen valve, by use of an alloy of 90 parts Zn and 10 parts Al. Heated to 50° the reduction is complete in 5 to 8 minutes if the TiO_2 is small, longer when more. Filter quickly into a flask containing $Fe_2(SO_4)_3$ and H_2SO_4 with a few grains of Na_2CO_3 to exclude air. It may prove desirable to maintain an atmosphere of CO_2 in the top of the funnel during filtration, but this is not considered

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absolutely necessary. The $FeSO_4$ reduced by the Ti_2O_3 is then titrated with standard permanganate. The use of a "reductor" tube for reducing was not found quite as convenient, though it may be made efficient.

Manganese in Presence of Iron—Colorimetric. Schmidt (*J. Am. Chem. Soc.*, XXXII., 965). Walters method modified. This depends on the formation of permanganate by persulfate in presence of $AgNO_3$.

A standard Mn solution may be prepared by use of *N*/10 permanganate. Run 182.1 c.c. of the *N*/10 solution into a liter flask, add 100 c.c. of 10 per cent. H_2SO_4, then 12 c.c. of H_2O_2 solution, warm to expel the O and then cool and dilute to the mark. 10 c.c. will contain 0.002 gm. Mn. Prepare also a solution of 20 gms. $AgNO_3$ per liter. 1 c.c. of this should be used for every mg. of Mn present.

Of the solution to be tested take 5 c.c.—dilute to about 100 c.c., add a few c.c. of ether and then precipitate with $(NH_4)_2S$. The ether assists in hastening the separation of the precipitate. Heat and stir vigorously, and when the precipitate granulates well, filter through a small filter and wash with diluted $(NH_4)_2S$. Dissolve through the filter with as little dilute H_2SO_4 as possible, boil out H_2S, add HNO_3 to oxidize the Fe, and then a drop of the $AgNO_3$. If a turbidity occurs add a slight excess of the $AgNO_3$ heat and filter into a 100 c.c. flask. The solution should now be perfectly clear. Add 1 c.c. $AgNO_3$ for every mg. of Mn supposed to be present, and then 1 gm. ammonium persulfate. Heat on the steam-bath. The color passes through yellow to the permanganate tint in about one minute, cool, fill to the mark, stopper and shake well. Treat 10 c.c. of the standard Mn solution in the same manner, omitting the separation by sulfide. By the use of comparison tubes, such as are used for colorimetric carbon tests in steels, the proportions of Mn may be determined.

In some cases the coloration may be modified by the presence of brown Mn oxides. In such cases, reduce entirely by SO_2 or a sulfite, add more $AgNO_3$ solution and another portion of persulfate, and warm up as before.

Manganese by Volhard Titration. Influence of HF. Müller and Koppe (*Zts. anorg. Chem.*, LXVIII., 160). In presence of HF the reaction is not quantitative, nor is it easy to distinguish the end point. On the other hand, the addition of HF in titrating Fe is not productive of error unless much F or much manganous salt is present.

Manganese and Oxalic Acid. Schröder (*Zts. Off. Chem.*, XVI., 270). The oxygen of the air does not oxidize $H_2C_2O_4$ unless Mn salts are present, but in their presence (also in presence of dissolved TiO_2) air will effect such oxidation. The action is not very rapid, but it has some bearing on analyses of pyrolusite, standardizing permanganate, etc., when the titration is slowly performed (5 or 10 minutes). It is much slower when the temperature is below 50° C.

Zinc—Volumetric. Grossmann and Hölter (*Chem. Ztg.*, XXXIV., 181). By using the $AgNO_3$ and KI indicator, Zn salts may be titrated in presence of NH_4Cl with standard KCN solution in the same manner as for Cu and Ni salts.

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Reagent for Zinc, Nickel and Cobalt. Alvarez (*Ann. Chim. anal.*, XV., 129). The reagent is made by dissolving 10 parts $CoSO_4$ or $CoCl_2$ in 100 parts of water saturated with SO_2. KCN is then added until the precipitate which forms at first is redissolved. SO_2 is also added to the solutions tested.

With $ZnSO_4$—orange red precipitate dissolving in excess to dark red. The dried precipitate is orange, when heated to expel H_2O it becomes violet.

With $NiSO_4$ yellow precipitate becoming green on warming. This also disolves in excess, giving a yellow—bleached by tartaric.

With $CoSO_4$ red precipitate dissolving excess, not readily bleached by tartaric.

Nickel or Cobalt—Volumetric. Jamieson (*J. Am. Chem. Soc.*, XXXII., 757). The reagent used is ferrocyanide; 20 gms. per liter gives a solution of which 1 c.c. = about 0.003 gm. Co or Ni. Of the solution to be tested, three equal portions, containing presumably about 0.1 gm. of Ni or Co, are to be taken in separate beakers. Add to each 10 c.c. of 10 per cent. $FeCl_3$ solution and 2 to 3 gms. citric acid, then NH_4OH until the solutions have a *faint odor* of NH_3. The $FeCl_3$ is of course omitted if the substance already contains much Fe. Dilute to 100 c.c. and warm up to 65° or 75°. Run in the ferrocyanide slowly with constant stirring. The precipitate takes an appreciable time to form at first. As the titration progresses take out a few drops on the stirring rod, put them on a paraffined white plate, and add a drop of dilute acetic. Stir well and take another few drops and treat them in the same manner. Note the burette reading, and call this pair "test 1." Run in a little more ferrocyanide and make "test 2" in the same manner, and so on. After 5 minutes, a greenish color develops in the tests when the end point occurred. The first of the three portions may be used to get an approximate result, the others for the accurate determination.

Cu, Zn or Mn, etc., precipitable by ferrocyanide, interfere.

In the case of steels, dissolve in HNO_3, remove Mn if present by boiling down and adding $KClO_3$, and after boiling out Cl, filter off, and test the filtrate in the manner described.

Nickel and Cobalt. Volumetric. Grossmann (*Ch. Ztg.*, XXXIV., 673). If the Co is not over 7 to 8 per cent. of the Ni, the KCN titration with AgI indicator will determine both together.—Add NH_4Cl, then render slightly ammoniacal, and titrate with standard KCN. With these conditions the Co reacts in the same manner as the Ni, and does not form cobalticyanide. In another portion of the solution, separate out the Ni by use of sugar and dicyanodiamidine. This precipitate is dissolved in dilute HCl, NH_4Cl and a slight excess of NH_4OH as before, and titrate with the KCN. Carnot's method for Co in the presence of Ni, depending on the formation of Co_2O_3 in alkaline solution by H_2O_2, and then treating this oxide with KI and titrating the I, was found to give correct results only when the Ni and Co were present in certain relative proportions.

Cobalt and Nickel. Volumetric. Rupp and Pfenning (*Chem. Ztg.*,

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XXXIV., 322). For Co the metal must be obtained as $CoSO_4$, in a solution free from acidity. The solution should contain 0.2 to 0.75 per cent. of the metal. Place in a burette and run into 5 (10, 25) c.c. of *N/2* KCN until a faint permanent turbidity is obtained, 1 c.c. of the KCN = 0.0059 Co. The same process may be used for Ni—preferably after adding 5 to 20 drops of 10 per cent. NH_4OH. Too much is fatal to accuracy, 1 c.c. *N/2* KCN = 0.007375 Ni. With Ni the process can be reversed. To a measured volume of the Ni solution add phenolphthalein, and run in the KCN solution until it comes clear, with a decided red color. Or a known amount (excess) of the standard KCN is added, and then the excess is titrated back with *N/2* acid, using methyl orange indicator.

Nickel in Steel. Grossmann and Schück (*Analyst,* XXXV., 247). Dissolve 0.5 to 2 gms. of the sample in aqua regia and evaporate to 10 c.c. Filter and add 15 gms. Rochelle salt for every 1 gm. of steel taken. If a yellow precipitate forms add a little NH_4OH to redissolve. If much Mn is present add a little hydrazine sulfate to prevent oxidation. Then add NH_4OH and about 40 c.c. of 20 per cent. NaOH, and then a cold 10 per cent. solution of dicyandiamidine sulfate. If much Ni is present, it is safe to filter as soon as the supernatant liquid becomes clear. With smaller amounts some time must be allowed. Filter through a Gooch, wash with cold dilute NH_4OH, dry at 120 to 130° C. and weigh. The precipitate is $NiC_2H_8N_4O_2$ from which the per cent. of Ni can be calculated.

Copper—Delicate Test. Uhlenhuth (*Chem. Ztg.,* XXXIV., 887). 0.5 gm. of 1:2 diamino-anthraquinone-3-sufonic acid is dissolved in half a liter of water and 40 c.c. of NaOH solution (Gr. 1.4) is added. This reagent gives a blue coloration in solutions containing Cu in such small quantities that it cannot be detected by the ordinary tests.

Cadmium Test. Wöhler and Hirschberg (*Ber.,* XLIII., 753). In presence of Cu after adding NH_4OH, the solution is decolorized by KCN, and then H_2S should give yellow CdS if Cd is present. It is noted that frequently when Cd is not present, a yellow precipitate is obtained, which however usually has more the color of Sb_2S_3. The authors find that this is an organic compound, $C_2H_4N_2S_2$ (Rubeanic acid). Low temperatures of the solution favor its formation; excess of KCN over and above that necessary to decolorize the Cu compound, hinders it.

Mercuric Chloride in Gun Cotton. Jaunopoulos (*Zts. ges. Schiess-sprengst. wesen,* III., 47). 30 to 40 gms. of the sample is kneaded to a paste with water, and 15 or 20 bits of clean bright Cu wire mixed in with it, about 3 c.c. conc. HCl being added. After standing several days the Cu is separated out, and washed successively with NaOH, water, alcohol and ether. The Cu is then placed in a dry glass tube and gently heated to obtain a sublimate, which may be microscopically examined or tested with I vapor.

Separating Bismuth from Lead. Little and Cahen (*Analyst,* XXXV., 301). Nearly twenty different methods have been proposed, very few of which have received general approval. The authors find

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that of Benkert and Smith to be satisfactory—double precipitation of the Bi as basic formate. Their procedure is: Dilute the solution to 300 c.c. (cold), add Na_2CO_3 until it is opalescent but contains no visible precipitate, add 10 to 15 c.c. of 10 per cent. $NaCHO_2$ slowly with stirring, and then 5 to 10 drops $HCHO_2$. A white precipitate settles out —bring to a boil, keep boiling for 5 minutes, settle—wash by decantation, and then on the filter with boiling water. Redissolve in as little as possible of hot dilute HNO_3, and repeat the precipitation. In the precipitate the Bi is estimated by Stähler's phosphate method and in the filtrate Pb is separated by H_2S, dissolved and determined.

Lead in Small Amounts. Vernon Harcourt (*J. Lond. Chem. Soc.*, XCVII., 841). The process consists in a colorimetric comparison of the depth of color obtained by H_2S. The solution containing the Pb should be acid, but alkali acetate should be added, also (in 50 c.c.) 10 c.c. of 50 per cent. sugar solution. Standard colors for comparison are best made by mixing solutions of $Fe_2(SO_4)_3$, $CuSO_4$ and $CoSO_4$ in the proportions found to be necessary by experiment. When obtained and diluted to the proper shades, the tubes should be sealed up, when the tint remains unchanged for an indefinitely long time. PbS, such as would be obtained with a standard solution of Pb, fades gradually in daylight, and cannot be kept.

The series of standard colors recommended should correspond with 0.00025, 0.00019, 0.00014, 0.00010, 0.00007, 0.00005, 0.00003, 0.000015 and 0.000005 gm. of Pb (or PbO, according to requirements).

Lead in Earthenware Glazes. Rogers (Report of Committee on Lead, etc., in Potteries). A few drops of HF are applied to the glaze. After allowing a certain period for action (40 seconds) the spot is removed by applying a bit of absorbent paper. The paper is then dipped in H_2SO_4 solution, and after washing off the soluble sulfates $(NH_4)_2S$ is applied, which gives a stain of PbS. By attention to uniformity in applying the test, a fairly accurate estimate can be made of the proportion of Pb in the glaze.

Lead—Volumetric. Oddo and Beretta (*Gaz. Chim. ital.*, XXXIX., 671). The Pb is titrated with standard K_2CrO_4 or $K_2Cr_2O_7$ in acetic solution, the end point being determined by "spot test" with diphenylcarbazide which gives a blue coloration with CrO_3 as soon as that is in excess.

Lead in Babbitt Metal and Similar Alloys. Mann (*Chem. Ztg.*, XXXIV., 917). In alloys of this kind small percentages of Pb are determined with difficulty, if enough of the alloy is taken to give weighable amounts. The difficulty is obviated by precipitation with H_2S in ammoniacal solution containing KCN.

Dissolve 10 gms. of the alloy in 45 c.c. HCl with 5 c.c. HNO_3 both conc. Add 10 gms. tartaric and dilute to 200 c.c. Cool, neutralize with NH_4OH and then add 100 c.c. more of NH_4OH (Gr. 0.91). Add KCN to decolorization, and then 5 gms. of solid KCN. Warm to 70° C. and pass in H_2S. Filter, and wash with a hot solution containing 50 gms. commercial $(NH_4)_2S$ and 10 gms. KCN per liter, which has been heated until it becomes colorless. The precipitate still contains Sn and

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Sb. Dissolve in HCl (Gr. 1.11) containing Br, and reprecipitate as before, using NH_4OH and KCN. Moisten this precipitate with H_2SO_4 and heat to dryness. Some Sn and Sb still adhere as oxides, but may be left behind if the residue is treated with boiling $NH_4C_2H_3O_2$ solution, which takes up the $PbSO_4$.

Arsenic—Separation. Jannasch and Seidel (*Ber.*, XLIII., 1218). As can be separated by distillation in presence of HBr or a bromide and hydrazine more rapidly than when the HBr is omitted. The material to be analyzed is rinsed into a small flask—80 to 100 c.c. of HCl (Gr. 1.19) added, then 1 gm. KBr or conc. HBr and 3 gms. hydrazine (chloride or sulfate). The receiver at the end of the condenser should contain 200 c.c. of water. On distilling for about an hour when the contents of the flask will have been reduced to 25–30 c.c. all of the As will be found in the distillate. The flask will contain all the other metals of the substance tested including Sb.

Arsenic. Krickhaus (*Eng. and Min. Jour.*, XC., 357). Mix 0.25 to 1 gm. of the sample with 3 to 4 gms. of ZnO mixture (3 parts ZnO to 1 part dry Na_2CO_3) (vid QUARTERLY, XXIX., 183) in a porcelain crucible. Cover the mass with a little of the ZnO mixture and heat highly in a muffle for 15 to 20 minutes. Cool, crush any lumps which may have formed, and boil with 50 c.c. of water, filter and wash, concentrating the filtrate and washings to about 50 c.c., and then add an equal bulk of strong HCl. When cool, add 10 c.c. of a 20 per cent. solution of KI, and then 100 c.c. of cold water. Titrate with standard $Na_2S_2O_3$. In the author's estimation the use of starch is unnecessary, the yellow color of the free I being sufficient to show the end reaction by its disappearance.

Arsenic, Antimony and Tin—Separation. Caven (*Proc. Chem. Soc.*, XXVI., 176). Modification of Walker's method. The mixed sulfides, after filtering and washing, are boiled with strong HCl, which leaves only As_2S_3 insoluble. When filtered off, this may be dissolved in $(NH_4)_2CO_3$, etc. The solution containing $SnCl_4$ and $SbCl_3$ when free from H_2S, is treated with KOH until the hydroxides just redissolve. Br is then added until the solution is permanently yellow, which affords antimonate. Solid NH_4Cl is then added cautiously at first, the KBrO reacting rather vigorously to afford N gas. Eventually on boiling Sn hydrates precipitate, leaving the Sb in the solution.

Arsenic, Antimony and Tin. Volumetric with Ferricyanide. Palmer (*Am. J. Sci.*, XXIX., 399). As, Sb and Sn, when in the *ous* form, are oxidized in alkaline solution to their respective *ic* forms by ferricyanide. The ferrocyanide resulting is titrated in the acidified solution by means of standard permanganate. It is necessary to add five times as much of the ferricyanide as would suffice for the oxidation with about 25 c.c. of 20 per cent. KOH solution. In the case of Sn enough KOH must be added to completely redissolve the SnO_2 produced. The bulk of the solutions must be kept as low as possible. Some minutes must be allowed for the reaction to take place, after adding the ferricyanide. Then in the case of As, about 10 gms. $(NH_4)_2SO_4$ and 100 c.c. magnesia mixture are added, the $MgNH_4AsO_4$ filtered off, and the solution acidified with H_2SO_4 before titration.

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Removal of the Sb is unnecessary, when that element is to be determined. With Sn, 10 gms. $(NH_4)_2SO_4$ are added, and the solution is warmed to 50 or 60° C. to precipitate out Sn. This is filtered off, and the filtrate acidified and titrated as with the others.

Antimony and Tin in White Metal. Schurmann (*Mitt. Kgl. Materialpruf.' amt.,* XXVII., 349). Treat 1 gm. of the turnings with 20 c.c. of chloroform and 5 c.c. of Br, heating if necessary. Keep warm for about 30 minutes, crushing any undecomposed particles with a glass rod. Cool, filter through asbestos, add 60 c.c. of 10 per cent. oxalic; shake well. After settling run off the chloroform and shake that up with 60 c.c. of 4 per cent. oxalic. Combine the two aqueous solutions, and expel Br by a current of CO_2. Dilute to 300 c.c., bring to boiling, and pass H_2S, which precipitates Sb. The Sn in the filtrate may be separated out by the methods usually applied in the Clark process.

Babbitt Metals, etc. Demorest (*J. Ind. and Eng. Chem.,* II., 80). *In the absence of Sb,* dissolve 2 gms. in conc. HNO_3 (Gr. 1.42) and evaporate to dryness to obtain a filterable form of the metastannic acid. Add 50 c.c. H_2O and 5 c.c. HNO_3; heat up and filter. Wash a few times, then rinse the precipitate back into the original beaker, and digest with 25 c.c. NH_4HS, poured through the filter. The SnS_2 dissolves leaving the PbS and Cu_2S which had adhered to it. Heat and agitate for about 10 minutes, and filter through the original filter. The recovered Pb and Cu are dissolved in HNO_3 and the solution added to the first filtrate. The NH_4HS solution of Sn with washings is brought to 200 c.c., of which 100 c.c. are taken, and 10 c.c. NH_4HS and 4 gms. KCN are added, and the solution electrolyzed with a 5 to 6 amp. current, which is stirred by a mechanical stirrer. Of the solution containing Cu and Pb an aliquot portion is electrolyzed in a similar manner as to current strength and stirring.

When Sb is present. Cover 1 gm. of the alloy with 20 c.c. H_2O in a beaker, add 5 gms. tartaric, dissolve it, and then add 10 c.c. conc. HNO_3. When dissolved dilute to 50 c.c. and run in strong NaOH until the precipitate which first forms is redissolved. *Do not heat* at this point. Pour slowly, with constant shaking, into a flask containing 150 c.c. of a boiling hot solution of 10 gms. NaOH with 10 c.c. of colorless saturated solution of Na_2S. Agitate for some minutes, then stand and settle. Decant through a ribbed filter and wash twice by decantation (25 c.c. of water each time). Redissolve the black sulfides by use of 5 c.c. HNO_3 and warming. Render alkaline with NaOH, and again precipitate with Na_2S, heating and shaking as before. Filter through the same filter and wash with hot water containing a little Na_2S. Dry the sulfides on the filter.

From the filtrate containing Sn and Sb, those metals are precipitated by acidifying with H_2SO_4. Settle, decant through a filter and then set the beaker under the filter and pour through it 25 c.c. of the NH_4HS solution diluted with 25 c.c. of water. Dilute to 150 c.c., add 4 or 5 gms. KCN and electrolyze as before. Platinum gauze should be used as the cathode. The current strength should be 8 to 10 volts and 5 or

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6 amp. The beaker is best set in a basin of water to prevent heating by the current. 30 to 60 minutes suffice. Wash the cathode with water, then with alcohol, dry and weigh combined Sn and Sb. Reconnect the cathode, run the current for 10 minutes, clean up and weigh again. If the weight has not perceptibly increased, the deposition can be regarded as complete. Run 5 c.c. of strong HNO_3 over the cathode, using a pipette and tilting the beaker about, so that the HNO_3 touches every part of the deposit. Rinse off the HNO_3 with water, and dissolve off the black stains, etc., which remain by the use of 60 c.c. conc. HCl. Unite the HNO_3 and HCl solutions and boil vigorously to decompose the HNO_3. When that is gone add $KClO_3$ in small amounts at a time, and boil, passing a current of CO_2 until the fumes no longer affect iodide-starch paper. Dilute to 150 c.c., and cool under the tap. Then add 4 gms. KI and titrate Sb with standard $Na_2S_2O_3$ solution. The PbS and CuS left on the filter can be dissolved in HNO_3 and determined by means which will suggest themselves to analysts.

Tin in Presence of Antimony. Sanchez (*Bull. Soc. Chim.*, VII., 890). The method is based upon these points: In strong HCl solution $FeCl_2$ alters $SnCl_2$ to $SnCl_4$, the end reaction being a persistent greenish yellow color in the solution. No such effects are produced on Sb solutions. Also metallic Al completely precipitates Sb in elemental form from its solutions, but reduces Sn no further than the stannous condition. The $FeCl_3$ solution is made by dissolving 10 gms. of Fe by hydrogen or 10.5 gms. of filings in HCl with the aid of $KClO_3$, boiling out Cl and diluting to 1 Liter. This is standardized by means of an HCl solution of 1 gm. Sn diluted to 100 c.c. The mixed sulfides obtained in the ordinary course of procedure are dissolved in HCl with addition of $KClO_3$, filtered, then boiled with metallic Al, the Sb filtered off, and the Sn then titrated.

Tin Ores. Gray (*J. Soc. Chem. Met. S. Africa*, X., 312). Examination of method. Only the wet methods give trustworthy results: The methods of effecting solution are described, and commented on.

1. Reduction by heating in H and then dissolving in HCl.—Slow and uncertain; re-treatment usually necessary.

2. Heating with Zn dust followed by treatment with HCl.—Satisfactory.

3. Mixing ore with Zn dust, and treatment with HCl.—Useless.

4. Fusing with Na_2CO_3 and S, leaching out, etc.—Tedious, dirty and slow.

5. Fusion with KCN, leaching with H_2O, then dissolving Sn in KCl. —Loss of Sn.

6. Fusing with KOH or NaOH, and dissolving the melt in HCl.—Good but not rapid.

7. Fusion with Na_2O_2, extracting with H_2O, and adding HCl.—Best.

8. Fusion with KHF, extraction with H_2SO_4.—Method not tried.

The method recommended is: Fuse 0.5 to 2 gms. of the finely pulverized ore with 4 to 6 times its weight of Na_2O_2 in an iron crucible, extract with water, put solution in an Erlenmeyer flask, and add excess of HCl. Heat to 90° C., add iron nails, and continue the reduc-

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tion for about an hour after the solution becomes colorless. Cool rapidly, add a fragment of marble, and when cold add starch, and titrate with standard I, standardized against pure Sn.

Antimony. Schulte (*Chem. Centr.*, 1909, I., 1741). A method is proposed based on the separation of elemental Sb from sulfantimoniate solutions by use of Mg. The results are said to compare favorably with those by electrolytic methods.

Volumetric for Antimony. Schmidt (*Chem. Ztg.*, XXXIV., 453). Examination of Methods. 1. *Iodometric*—titration with standard I in presence of $NaHCO_3$. The end point is not very clear. If addition is stopped on the appearance of a pinkish tint the results are fairly good (within 0.2 per cent.). Sn interferes.

2. *Potassium bromate.* $KBrO_3 + 2HCl + 3Sb_2O_3 = 2KCl + 2HBr + 3Sb_2O_5$. The titration is performed in presence of dilute HCl. Methyl orange is used as indicator, the bleaching out of the pink color being the end point. The solutions should not be too dilute. With that precaution, the results are very exact. Sn also interferes in this method.

3. *Permanganate.* HCl is added in sufficient amount to redissolve the precipitate first formed and the solution is then titrated cold. Accuracy within 0.1 per cent. Sn does not interfere. 5.27 gms. $KMnO_4$ per liter gives a solution of which 1 c.c. = 0.01 gm. Sb.

Antimony Determinations. Beckett (*Ch. News*, CII., 101). Examination of methods. The conclusions are: The most accurate method is by precipitation as Sb_2S_3 by the method of Vortman and Metzl. (Precipitation in a boiling solution containing 20 per cent. conc. HCl.) Sb_2S_3 precipitated from HCl solution invariably contains some Cl not removable by washing. The amount can be reduced to 0.15 to 0.3 per cent. by heating to 300° in an atmosphere of CO_2.

Sb can be determined by titrating Sb''' compounds with I in presence of excess of $NaHCO_3$ (end product Sb^v) or by adding KI and HCl to Sb^v compounds and titrating the I set free by $Na_2S_2O_3$. In either of these cases the results are 1 per cent. too low, unless the Sb is calculated as having a molecular weight of 122. Determination as Sb_2O_4 (SbO_2) is untrustworthy. The product always contains a little more O than corresponds to the formula, and is also hygroscopic. Determination as Sb_2O_5 is untrustworthy.

Gold or Silver—Detecting small amounts. Armani and Barboni (*Zts. Chem. Ind. Kolloide*, VI., 290). The method is especially suitable in testing plated articles. The production of colloidal forms of the metals by use of formaldehyde is the principle brought into play.

For *silver*, treat the article with 1 part HNO_3 conc. (Gr. 1.41) and 9 parts H_2SO_4 of 66° Be. This dissolves off Ag but does not attack brass. A few drops of the solution are taken out, diluted with 2 to 3 c.c. of water, nearly neutralized with some 20 per cent. KOH, and then a mixture of 2 vols. 30 per cent. formaldehyd with 1 vol. of 20 per cent. KOH is added. Ag gives a violet color.

For gold treat with HNO_3. Dissolve the insoluble residue in aqua regia, dilute, filter if necessary, evaporate with H_2SO_4 to expel Cl

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and N oxides, dilute with water and add the mixture of KOH and formaldehyde as above. Au gives a fine violet coloration.

Assaying Gold Alloys. Smith (*J. Inst. Metals,* III., 98). For accurate work run a preliminary assay, to get the approximate composition. Then conduct the assay using test lead in these proportions.

For 22 carat, 8 parts Pb to 1 of Au; for 18 carat 15 parts Pb; for 12 carat 20 parts; for 9 carat 24 parts. Also run a check assay on a test piece made by weighing out the metals, to show the necessary correction.

Thorium in Monazite. Meyer and Speter (*Chem. Ztg.,* XXXIV., 306). Heat 50 gms. of the sample with 100 c.c. conc. H_2SO_4 to 250° C. for six hours. Cool, and add 500 c.c. cold water. Filter, cool, dilute to 1 liter and take 100 c.c. for the analysis. Add 50 c.c. conc. HNO_3, cool, and add a cold solution containing 15 gms. KIO_3, 50 c.c. HNO_3 and 30 c.c. of water. After half an hour filter and wash with a solution containing 2 gms. KIO_3, 50 c.c. dilute HNO_3 and 200 c.c. water. Then by means of a jet of hot water, transfer the precipitate back to the beaker, add 30 c.c. of strong HNO_3 and redissolve by boiling. Reprecipitate by addition of 4 gms. of KIO_3 and filter through the same filter. It will now be free from Cl. Wash as before. Dissolve in HCl with the aid of SO_2, precipitate boiling hot with NH_4OH. Wash with boiling water, redissolve in HCl and precipitate with excess of $H_2C_2O_4$. Wash with dilute HCl, ignite and weigh.

Tungsten in Steels. Bartonec (*Oesterr. Chem. Ztg.,* XII., 114). After dissolving in aqua regia the solution must be evaporated low, but not to dryness as that makes it impossible to separate the Fe. For gravimetric determination, after washing off all Fe by dilute HCl dissolve through the filter with $(NH_4)_2CO_3$, evaporate, etc. For volumetric, wash with dilute HCl, then with 5 per cent. $NaNO_3$ solution to prevent colloidal WO_3. Dissolve through the filter with 30 c.c. *N*/10 NaOH, and wash through with hot water. Na_2WO_4 forms. Titrate back with *N*/10 H_2SO_4. With phenolphthalein the end point occurs at the ratio $2NaOH : WO_3$. With methyl orange at the ratio $2NaOH : 4WO_3$.

Vanadium—gravimetric. Browning and Palmer (*Am. J. Sci.,* XXX., 220). In acid solution $AgNO_2$ gives precipitates of varying composition, but in one perfectly neutral, the composition of the precipitate is constant. The V_2O_5 solution is acidified with HNO_3 then rendered slightly alkaline with NH_4OH, and the excess boiled out. The color of the solution serves as a guide. When alkaline it is yellow, when nearly neutral it is colorless, and when neutral it shows a pale yellow tint again. While boiling an excess of $AgNO_3$ is added, and the solution vigorously stirred to favor coagulation of the precipitate. Filter through a Gooch (asbestos) wash, and ignite at a temperature below its melting point before weighing.

Molybdenum, Vanadium, Selenium and Tellurium. Determination by silver. Perkins (*Am. J. Sci.,* XXIX. ——). A process has already been described for gravimetric determination of free I by use of electrolytically prepared Ag. (*Am. J. Sci.,* XXVIII., 33.) The process has also been applied to the estimation of combined I and other

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halogens, also oxidizers which react definitely with KI. (*Am. J. Sci.*, XXIX., 338.) In this paper the process has been applied to some of the rarer elements. The solutions of molybdate vanadate, selenite or tellurite were shaken in flasks with KI, HCl and the Ag in an atmosphere of H, and the weight of AgI + surplus Ag weighed up.

MoO_3 was found to form MoO_2I with evolution of I.

V_2O_5 was converted to V_2O_4, liberating I_2.

SeO_2 and TeO_2 were reduced to elemental form, each evolving I_4.

In these cases the Se and Te was also weighed up with the mixed AgI and Ag.

Indium. Thiel and Koelsch (*Zts. anorg. Chem.*, LXVI.). In may be determined quantitatively by precipitation with NH_4OH, and heating to 850° in an electric furnace. A Gooch crucible should be used. The product is In_2O_3. Overheating will cause a loss of O and consequent errors. Strong NH_4OH dissolves In_2O_3 or $In(OH)_3$, a fact which may be utilized in certain separations.

Palladium. Gutbier and Falco (*Zts. anal. Chem.*, XLIX., 287). The material containing not over 0.1 gm. Pd is dissolved in very dilute HCl diluted to at least 150 c.c., brought to a boil and a boiling solution of hydrazine sulfate is added. Then stand on the boiling water-bath for 20 minutes. A mirror of the metal forms at first on the glass, but is later dislodged by the gas evolution. Boil 30 minutes, cool slightly, filter and wash off all HCl. Burn the paper separately. Ignite first in air, then in H, finally in CO_2 containing no O; repeat until the weight is constant. In dilute NH_4OH solution a similar procedure will also afford good results.

Thallium—New Reagent. Tanatar and Petroff (*J. Russ. Phys. Chem. Soc.*, XLII., 94). The reagent is sodium cobaltinitrite, $Na_3Co(NO_2)_6$, which is used in cold slightly acid solution. The precipitate of $Th_3Co(NO_2)_6$ is red, crystalline, insoluble in hot water or cold dilute acids. Solubility in H_2O about 1 in 10,000. Pb is the only other metal giving a precipitate with this reagent, and that is soluble in hot water.

Tellurium Determination. Browning and Flint (*Zts. anorg. Chem.*, LXIV., 104). The method is similar to that used for Bi—precipitation by large dilution of BiOCl. The solution should contain not more than 0.2 gm. TeO_2. Dissolve the substance in 10 per cent. KOH. (2 c.c. suffices for 0.2 gm. TeO_2.) Acidulate slightly with HCl, and dilute to 200 c.c., with boiling water. The TeO_2 precipitates in fine crystals. Add a very slight excess of NH_4OH, and then acidify with acetic. Let stand 12 to 24 hours, and filter, washing with cold water. The TeO_2 is insoluble in cold water, alcohol, acetic acid, and cold 10 per cent. solutions of $NH_4C_2H_3O_2$. The method can be used for separating from Se which does not interfere.

Iodide in Presence of Bromide or Chloride. Bray and MacKay (*J. Am. Chem. Soc.*, XXXII., 1193). The method consists in setting free I by $KMnO_4$ in dilute H_2SO_4 solution, dissolving in CCl_4 and titrating the I with standard "hypo." For the purpose the CCl_4 must be first purified by shaking with I and dilute H_2SO_4, allowing to stand for a

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few hours, then drawing off by means of a separator funnel, decolorizing by use of $Na_2S_2O_3$ solution, and washing thoroughly with successive portions of water.

The $Na_2S_2O_3$ solution should be $N/10$ or $N/5$ strength and the $KMnO_4$ should nearly correspond with it.

The process is as follows: Put 25 to 30 c.c. CCl_4 in a separator funnel of about 250 c.c. capacity. Then add solution of a known amount of the substance, and acidify with a few c.c. of dilute H_2SO_4, limiting the amount to about 5 c.c. of acid (Sp. Gr. 1.2) if bromides are present. Make the volume of the H_2O solution 75 to 100 c.c. Now run in some $KMnO_4$ solution from a burette until apparently the major part of the I has been freed. Stopper, and shake well to cause the transferrence of the I to the CCl_4. Let stand half a minute. Run off the CCl_4 into a stoppered flask containing 10 per cent. KI solution acidified with acetic. Add more CCl_4 and repeat the treatment with $KMnO_4$ solution until the I is all removed and the solution shows a slight pink tinge due to $KMnO_4$. Bromides are but slowly acted upon by $KMnO_4$, and by working in this manner, only I will be set free. Then titrate the free I in the flask with standard $Na_2S_2O_3$, shaking constantly, especially when near the end point. Direct titration, using $KMnO_4$ solution alone, was found to give a slight error of 0.4 to 0.5 per cent. under variations of the conditions.

Methylene Blue as Iodometric Indicator. Sinnatt (*Analyst,* XXV., 309). Adding methylene blue to a solution of I in KI gives a precipitate. In dilute solutions, the blue changes to yellowish green, and finally clear yellowish brown. For indicator purposes dissolve 0.05 gm. methylene blue in 1 liter. Use about 1 c.c. for each 50 c.c. of solution titrated.

Sulfur in Illuminating Gas. Somerville (*J. Gas-lighting,* CXII., 28). The gas is burned in a Bunsen burner at the rate of half a foot per hour, and the products of combustion are led through a known amount of standard I solution containing a specially prepared starch indicator. When decolorization ensues, a definite amount of S in the gas has been burned to SO_2. It only remains to read the meter to determine how much gas contained that amount.

Sulfur for Agricultural Uses. Bruno (*Ann. Falsif.,* III., 129). Three forms occur in the market, which should contain as follows:

	Fine S.	Ash.	Moisture.
1. Flowers (sublimed)	98.5	0.5	0.5
2. Powdered	97	2	0.5
3. Precipitated	within 2 per cent. of guarantee.		

For moisture dry 5 gms. at 90 to 95° C. for 2 hours; burn off and weigh up ash.

Boil 1 gm. of the sample in a flask with 10 c.c. NaOH of 36° Be and 20 c.c. of water. After boiling 5 minutes rinse the S down from the sides of the flask and boil 15 minutes longer—cool—dilute to 100 c.c. Take out 25 c.c., bring it to a boil and add five successive (5 c.c.) lots of hypobromite (made with 2 c.c. Br 10 c.c. strong NaOH and 20 c.c. water). Acidify decidedly with HCl—boil out excess of Br,

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cool, dilute to 100 c.c., and in measured portions of this solution determine S by $BaCl_2$ precipitation. When "Flowers of Sulfur" are freshly sublimed about 33 per cent. is insoluble in CS_2. As time elapses, the solubility increases until about 12 per cent. only may remain insoluble.

Sulfur in Pyrites and Marcasite. Allen and Johnston (*J. Ind. & Eng. Chem.*, II., 196). A source of error apart from the wet treatment consists in the fact that dry grinding of these minerals always affords some $FeSO_4$ which is the larger, the longer the grinding is continued. In an extreme case it is calculated that the error would amount to 0.8 per cent of the S present. The process recommended for an umpire or control method, is oxidation by the method of Carius (HNO_3 in a sealed tube) precipitating out Fe twice by Na_2CO_3 (NH_4OH is objectionable) and then precipitation by $BaCl_2$ in a boiling solution but slightly acidified with HCl. Three corrections should be applied. (1) for the solubility of $BaSO_4$. This depends on the volume of solution and the proportion of free HCl. (2) For occlusion of Na_2SO_4: depends on the proportion of alkaline chloride present. (3) Free H_2SO_4 lost on ignition: depends on amounts of free acid and of NaCl in the solution, also the rate at which the precipitation is made. Apparently caused by the fact that $NaHSO_4$ is occluded by the precipitate.

The sum of these errors should not be over 0.2 per cent. of the total S.

Sulfates—in refined potassium salts, etc. Roemer (*Zts. anal. Chem.*, XLIX., 490). A modification of the volumetric chromate method. A *N/2* $BaCl_2$ solution is prepared, as also a $K_2Cr_2O_7$ solution which exactly precipitates it c.c. for c.c. A weighed quantity of the salt to be tested, possibly 50 gms., is dissolved in hot water, some HCl added, and 10 c.c. of the $BaCl_2$ solution for every 0.4 per cent. SO_3 supposably present. After boiling, the $K_2Cr_2O_7$ solution is added, in exactly the same amount as the $BaCl_2$ and the solution is made slightly alkaline. Boil, cool, make up to 200 c.c., filter off 100 c.c., which now contains chromate exactly corresponding to the SO_3 in the original, and titrate with $FeSO_4$ or with KI and $Na_2S_2O_3$.

Barium Sulfate. Van't Kruys (*Zts. anal. Chem.*, XLIX., 393). $BaSO_4$ invariably carries with it other sulfates, which constitute the chief source of error in the determination of S, SO_3, etc. $CaSO_4$ especially causes this trouble. But long boiling of the precipitate so contaminated with much HCl and $BaCl_2$ will afford pure $BaSO_4$ which is not true as regards other sulfates when contaminants (such as Na_2SO_4, $MgSO_4$ etc.) are present.

The author proposes, therefore, that to the solution be first added as much $CaCl_2$ as is doubly sufficient to combine with all the SO_4 present. A 10 per cent. hot solution of $BaCl_2$ is then slowly added. Then an equal bulk of conc. HCl, and the mixture is kept at near the boiling point for 3 hours. The supernatant liquid is decanted off, and evaporated to recover the small amount of $BaSO_4$ dissolved. The precipitate is digested for another 3 hours with strong HCl (1:1) containing some $BaCl_2$. The most of this supernatant liquid is decanted and evaporated with the first lot down to very small bulk. Then by use of

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about 100 c.c. of water, rinse the whole back into the precipitation vessel, boil half an hour and filter, wash, etc.

Ferro Silicon. Preuss (*Zts. angew. Chem.*, XXIII., 201). Fusion of 0.5 gm. with 10 gms. Na_2CO_3 and 0.5 gm. Na_2O_2 in Pt crucible was one method tried. In another 0.5 gm. of the sample was fused with 10 gms. KOH in a Ni crucible. The melts were leached with water, evaporated, etc., and gave concordant results. Preference is given to the KOH method because cheaper. Samples containing up to 90 per cent. Si were analyzed. For samples containing about 10 per cent Si, it is recommended to heat for an hour with 10 gms $KClO_3$ and 100 c.c. conc. HCl. Evaporate and fuse the residue with KOH, treating in the manner above indicated.

Boric Acid. Jannasch and Harwood (*J. f. prakt. Chem.*, LXXX., 134). Preliminary communication. A current of CCl_4 vapor does not carry B_2O_3 with it, but if the CCl_4 receives one fourth its volume of $CHCl_3$, the B_2O_3 will be carried along with the vapors.

Phosphoric Acid—Detection in Minerals. Lidoff (*Zts. anal Chem.*, XLVIII., 172). The mineral is mixed with metallic Mg and Al in an iron crucible, and heated to redness. After cooling add a 20 per cent. solution of KOH. The gases evolved are filtered through cotton, and then passed over a paper soaked in a 5 per cent. solution of $Cu(C_2H_3O_2)_2$, acidified with acetic. PH_3 gives a black stain which disappears by oxidation in a current of moist air. Bo, Si, Sb and S give off no hydrides, AsH_3 gives an inoxidizable stain readily distinguished from that made by the PH_3. (This test was long ago used by Bunsen, who depended only on the characteristic odor of PH_3 for the detection of P.)

Carbon in Iron and Steel. Kohout (*C. News*, CII., 23). The correspondent states that he has found the addition of $KClO_3$ to the double chloride solution is of great advantage, especially in the matter of speed. His solution is made up thus:

K_2CuCl_4	560 gms.
$KClO_3$	5 gms.
HCl	250 c.c.
Distilled water	1,600 c.c.

250 c.c. of this solution, with frequent stirring will dissolve the iron of 2.73 gms. of steel containing 1 per cent. C in about 15 minutes.

Carbon in Iron and Steel. Trembour (*Chem. Eng.*, XI.). The method described is that of direct combustion in a current of O. If the drillings can be brought to a degree of communition that they will pass a 20-mesh sieve, they will burn readily if mixed with pure Fe_2O_3. If larger, the addition of some powdered pure Fe is necessary. The author prepares pure Fe_2O_3 by evaporating and heating Fe_2Cl_6 solution. From this pure Fe may be prepared by heating in a current of H. Most of the "Iron by Hydrogen" on the market contains some C.

A fused silica combustion tube is better for the purpose than the porcelain combustion tube. 1 to 3 gms. of the sample are mixed with 1½ to 4½ gms. Fe_2O_3 to which (if coarse) 1 to 2 gms. of powdered Fe is added. The mixture is spread upon a layer of Fe_2O_3 in a porce-

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lain boat. The combustion tube, connected with the requisite absorption tubes (potash bulbs or other) is heated to full heat except at the spot where the boat is to be placed. The boat is then introduced, and after making the connections with the O supply, etc., the heat is applied to the boat. The reaction takes but one or two minutes. The material remaining in the boat after cooling should show no effervescence with acid.

Hydrocyanic Acid—Colorimetric by Picric Acid. Chapman (*Analyst,* XXXV., 469). Isopurpuric acid is formed when the picric acid is in excess in fairly concentrated solutions, neutral, and at elevated temperatures. Under other conditions there is a formation of more or less of the amino-nitro-phenoxides (picramic acid and others). Aldehyd, acetone, H_2S and possibly furfural give the same coloration, hence the test must be used with some discretion. Under the conditions used by the author, the test was found to give proportional colorations only when the HCN per 100 c.c. was between 0.002 and 0.008 gm.

Cyanide in Small Amounts. Qualitative. Lockemann (*Ber.,* XLIII., 2127). The use of the Prussian blue, and sulfocyanate tests is recommended. The material is treated in a test-tube with dilute H_2SO_4 and the tube partially immersed in the water of a boiling water bath. Across the mouth of the tube is stretched a piece of filter paper, moistened with solution of fixed alkali (for sulfocyanate test, of Na_2S). After a few minutes remove the paper. For the Prussian blue test, moisten with very dilute $FeSO_4$, allow to oxidize in the air, and then add a drop of HCl. For the sulfocyanate test moisten with dilute $FeCl_3$ and then apply a drop of HCl.

Cyanides—Colorimetric. Berl and Delpy (*Ber.,* XLIII., 1430). The Prussian-blue test can be made colorimetrically quantitative if care is exercised in the use of the reagents. Render alkaline with KOH(1:1), avoiding too large an excess, which is detrimental. Starting from neutrality two drops will ordinarily suffice. Add $FeSO_4$ solution (1:30) containing some ferric salt. The proportion used must afford at least 2 mol. $FeSO_4$ to 1 of HCN. Allow to stand at least 10 minutes with frequent agitation, then boil 2 to 15 minutes—proportional to the amount of HCN present, cool completely and acidify with 10 per cent. HCl. The color develops slowly. After 5 hours standing the color can be shaken up after diluting to 100 c.c., and will remain suspended long enough for color comparisons.

Cyanide—Rapid Commercial Analysis. Bell (*Eng. & Min. J.,* LXXXIX., 1114). SiO_2, metals, sulfates, etc., are separately estimated if required. Moisture is obtained by loss on fusion. For Na and K, a clear solution of a known amount is evaporated down with HCl, the chlorides weighed and then the combined Cl determined. Since NaCl contains 60.6 per cent. Cl and KCl contains 47.6 per cent. Cl each one per cent. of Cl in the mixed chlorides above 47.6 per cent. represents 7.69 per cent. NaCl out of the total, the remainder being KCl. The CN is estimated, and all K is calculated to KCN, the remainder to NaCN. Any surplus Na is reckoned as Na_2CO_3 unless Cl has been determined.

Ferro- and Ferricyanides. Iodometric. Mecklenberg (*Zts. anorg.*

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Chem., LXVII., 322). For ferricyanide dissolve in 500 to 750 c.c. of water, and add 10–15 c.c. conc. HCl, 10–20 c.c. (10 per cent.) KI solution and 10 c.c. of *2N* $ZnSO_4$. Let stand in a closed flask for 3 minutes, then add a little starch solution and titrate with *N*/10 $Na_2S_2O_3$. Carry the titration nearly to completion, then allow to stand 3 minutes, and then finish. 1 c.c. *N*/10 $Na_2S_2O_3 = 0.032921$ K_3FeCy_6. The $ZnSO_4$ precipitates the ferrocyanide as fast as it forms. The KCl prevents the formation of double KZn ferricyanides.

Ferrocyanide may be oxidized by $KMnO_4$ to ferricyanide, and then titrated in this manner, the excess of $KMnO_4$ being destroyed by oxalic. Or, $NaC_2H_3O_2$ may be added to a solution made up as before (omitting the $ZnSO_4$) and a measured excess of standard $KMnO_4$ is added. The amount of excess is determined by adding KI and titrating back with *N*/10 $Na_2S_2O_3$. In the absence of $ZnSO_4$, the ferricyanide does not react with KI.

Nitrogen in Organic Substances. Russell Ellis (*Ch. News*, CII., 187). Castellana proposes to heat Na_2CO_3 or K_2CO_3 with Mg, until action commences, and then to add the substance to be tested. The result is the formation of cyanide, proven by prussian blue test. The author finds that in conducting the test the Mg partially forms a nitride with the nitrogen of the air, and consequently cyanide is always formed. The test is therefore useless.

Nitric Acid and Nitrates. Qualitative. Schmidt and Lumpp (Ber., XLIII., 794). The reagent consists of 0.1 gm. of Di—(9.10 monoxyphenanthryl) amin per liter of conc. H_2SO_4. This has a blue color but with nitric acid or nitrates shows wine red. Dilution prevents the reaction so the substance tested must be in solid or very concentrated form. 2 to 3 c.c. of the reagent are used for a test on a granule of the material. Nitrates, unless in large amount, do not give the reaction. Chromates, chlorates, etc., give colors but they differ positively from that given by nitrates.

Nitrates by Phenylsulfonic Method. Stewart and Graves (*J. Am. Chem. Soc.*, XXXII., 756). The effect of chlorides upon the results obtained by this method applied to waters, has been examined by many. The authors find that 0.1 part of nitric nitrogen can be correctly estimated in presence of Cl in chlorides up to about 2 parts (per million); beyond that point the results were low—approximating 90 per cent. of the correct result.

"Nitron"—Suggestions as to modes of management. Hes. (*Zts. anal. Chem.*, XLVIII., 81). In using this reagent as a precipitant for HNO_3 (vid QUARTERLY, XXVI., 425) the solution should contain about 0.1 per cent. HNO_3. Solutions more dilute are incompletely precipitated. Dextrine, gelatine, peptone and some other organic compounds prevent the formation of a crystalline precipitate: Al, Mg or NH_4 sulfates, as also $MgCl_2$ and KH_2PO_4, besides citric, tartaric, glucose or saccharose, exercise no disturbing effects. Oxalic causes irregular results, usually too high.

$HClO_3$, when present to the extent of 0.25 per cent., is also precipitated quantitatively by "nitron." By suitable management, both $HClO_3$ and HNO_3 can be determined in the same solution by use of this reagent.

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Ganghofer (*ib.*, p. 545) finds that the precipitation in presence of dextrine gives correct results if a considerable time is allowed for the precipitation. That gelatine does not interfere unless in large amount, when it can be removed by tannin, removing the excess of tannin by neutral Pb $(C_2H_3O_2)_2$ solution.

Nitric Acid—Volumetric. Bronnert (*Bull. Soc. Ind. Mulhouse,* LXXX., 254). The method is intended chiefly for the examination of nitrocellulose and similar compounds, and is not expected to give very close results. (2 per cent. was found to be the maximum variation.) The process consists in titration with *N*/10 solution of $SnSO_4$ or $K_2Sn(SO_4)_2$ (sufficient HCl being added to clear it) in strong H_2SO_4 solution, using diphenylamine as indicator. The reaction is:

$$4SnSO_4 + 4H_2SO_4 + 2HNO_3 = 4Sn(SO_4)_2 + 5H_2O + N_2O.$$

To standardize, 0.15 gm. of thoroughly dried KNO_3 is dissolved in as little water as possible, and 50 c.c. conc. H_2SO_4 are added all at once to avoid heating so far as practicable. A drop of diphenylamine solution is added, and the solution is titrated with the standard solution until colorless. The first appearance of complete decolorization is taken as the end point. The standard solution is liable to alteration, and should be frequently restandardized. Keeping it cool is of advantage in prolonging its usableness.

Active Oxygen in Washing Powders. Bosshard and Gwicky (*Zts. angew. Chem.*, XXIII., 1153). Perborates are now frequently used in soap powders. The determination by titration methods is difficult, and probably never accurate. The authors find that by mixing the powders with MnO_2 and treating with dilute H_2SO_4, measurement of the evolved O gives good results. If carbonates are present, the CO_2 must of course be absorbed by NaOH before measuring. By suitable arrangement of the apparatus the CO_2 can be determined in the same operation.

Linoleum Analysis. Fritz (*Chem. Rev. Fett u. Harz Ind.*, XVII., 126). Heating the comminuted material in a sealed tube to 150° C. with C_6H_6 will extract much of the "linoleum cement." A subsequent extraction with a mixture of C_6H_6 and $CHCl_3$ will then remove it completely.

Petroleum Testing. Herr (*Chem. Ztg.*, XXXIV., 893). Nastukoff has devised a method for the determination of C_6H_6 and certain other hydrocarbons by the use of formaline. The author finds the method unsatisfactory and proposes to use methylal instead. Both act in the same manner, forming condensation products separable by the use of certain solvents. The reaction with methylal depends on the temperature at which the operation is conducted. When kept cold an oil yielded 7 per cent. of the product, whereas when worked hot 26 per cent. of a condensation product (of different composition) was obtained. A rapid test for detection and approximate estimation of tar oils in benzine, etc., consists in dissolving 2 c.c. of the oil and 4 c.c. methylal in 10 c.c. of light petroleum (b.p. 50°) in a graduated cylinder, adding 2 c.c. conc. H_2SO_4 drop by drop, stoppering and shaking for 1 minute. Settle one-half an hour, and then read off the volume of the precipitate.

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BOOK REVIEWS.

A Manual of Practical Assaying. By the late H. VAN F. FURMAN, E.M. Seventh edition, revised and enlarged. By WILLIAM D. PAIDOE, B.S., A.M. New York, John Wiley and Sons, 1910. 8vo. viii + 530 pages. Price 3.00.

The writer had occasion to review the sixth edition of the above book in the SCHOOL OF MINES QUARTERLY (Vol. XXX., No. 3, April, 1909), and most of the statements made therein will apply to the present edition, as ninety-five per cent. of the text remains unchanged.

In the present edition chapters on carbon, silica, lead, copper, cobalt and nickel, vanadium, tungsten, tin, arsenic, antimony, bismuth and sulphur have been partially rewritten, added to or added. This includes improvement in some of the old methods, the elimination or substitution of some of the less accurate methods for those of more recent date, and the addition of methods new to this book.

The selection of new methods seems to have been a matter of chance, for in some cases they are excellent, while in others they are most deplorable. As an example of the first case, the addition of the permanganate method for copper, which is gaining great favor in the assay of low grade copper ores.

To illustrate the second case, the introduction of the cyanide-mercuric-oxide method for nickel (one of the poor methods) is most absurd, when the dimethylglyoxime method has been proven to be infinitely more accurate and rapid.

Without question the book has been improved, but this improvement, compared with the possibilities, is so small that it is inconsiderable.

E. J. HALL.

Smoley's Tables: Parallel Tables of Logarithms and Squares. By CONSTANTINE SMOLEY, C.E. Sixth Edition, revised and enlarged. New York, The Engineering News Publishing Co.; London, Archibald Constable & Co., Ltd. Flexible morocco; 4¾ × 7 in.; 500 pp. $3.50.

Ordinarily all dimensions in structural designs are given in feet, inches and fractions of inches. To compute the dimensions of inclined members it is necessary to reduce these dimensions to feet and tenths in order to use logarithms. This well known book of tables "for Engineers, Architects and Students" is divided into two parts. Part I. gives, first, a table of the logarithms and squares of feet, inches and fractions of inches varying by 32ds of an inch from zero to 50 feet by 16ths from 50 to 100 feet. By the use of this table dimensions may therefore be used as arguments and the answers obtained in the required units without either reduction or interpolation. This is not only a great saving of time and labor but eliminates the danger of mistakes due to the reductions necessary by the ordinary method.

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The second table is also a great labor saver to those engaged in structural design work inasmuch as it gives the angles, logarithmic sives, cosines, etc., for triangles having a base of 12 inches and slopes up to 12 inches varying from 32ds of an inch.

Table 3 is a multiplication table for rivet spacing giving the feet, inches and fractions for any number of rivets up to 50 with spacings varying by 8ths from 1 to 6 inches. A table of decimal equivalents in feet for inches and fractions varying by 32ds completes Part I.

The second part of the book contains five place logarithm tables for numbers from 1 to 10,000 with proportional parts; logarithmic functions of angles for single minutes with proportional parts for seconds; natural functions of angles for each minute; and a table giving the circumferences and areas of circles (number = diameter), squares, cubes, square roots, cube roots and reciprocals for numbers from 1 to 1,000.

On the whole the sixth edition is a great improvement over the former editions not only in the completeness of the tables but also in the binding. The tables themselves are models of the art of tabulation and show how much the simplicity and clearness, and thus the usefulness, of any table depends on a proper selection of type and spacing as well as the omission of unnecessary details.

The *Engineering News* has published another of those model engineering publications which is worthy of the high standard which they have always held in this line.

J. K. F.

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THE

SCHOOL OF MINES

QUARTERLY.

VOL. XXXII. JULY, 1911. No. 4.

COPPER SMELTING IN MICHIGAN.

BY HENRY D. CONANT.*

Copper smelting in the Lake Superior district of Michigan comprises the melting of concentrates and mass containing native metallic copper, the refining of the copper, and the smelting of the slag formed during the melting operation to recover the high metal values carried by it. In the simplicity and directness of the methods and processes required the Lake practice differs very greatly from that of any other district and is altogether different from that involved in the treatment of the ores of copper. The absence of roasters, calciners, matte furnaces and converters, and the electrolytic refinery, essential to the smelting of copper sulphides—the usual source of copper in other districts—is apparent at once to the metallurgist; and the very simplicity of the operations has given rise to the widely spread idea that "there is no metallurgy in Lake Superior smelting." Yet as, according to Professor Egleston's definition of metallurgy, it is the science of making money out of smelting, the metallurgist will find plenty of opportunity to draw on his technical training. The effective use of fuel, the efficiency of labor, the avoidance of excessive amounts of metal in the reverberatory slag, the successful smelting of the slag and the prevention of losses in the waste slag present problems that require careful study; while the operation of refining, for many years regarded as a deeply mysterious proceeding, demands care and technical knowledge to handle the rude methods

* Superintendent, Lake Superior Smelting Co. E.M. Columbia, 1886.

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possible in such a manner as to obtain the most satisfactory results.

Copper occurs in the Lake district practically altogether in native form, either disseminated through conglomerate or amygdaloid rock in sizes varying from the finest particles to pieces weighing several pounds, or existing in masses running from a few pounds in weight up to several tons. After being hoisted from the mines the rock containing the copper is crushed, stamped and concentrated on jigs and slime tables until the percentage of the metal is brought from its initial figures in the vein rock of from one to three per cent. to from sixty to eighty according to the system best adapted to each kind of rock. The mass copper is broken and cut, and cleaned of rock as far as possible. The concentrates, known locally as mineral, and the mass are then shipped to the smelters ready for the furnace treatment.

The mineral and mass are generally, with some exceptions to be mentioned later, charged into reverberatory furnaces and melted down without any fluxes, the slag skimmed off as fast as it is formed and the remaining copper either refined in the same furnace or in a secondary reverberatory furnace into which it is tapped. The refined copper is then either ladled or dipped into moulds to form the several shapes in which it is marketed. The slag is skimmed usually into conical pots, some metallic copper settling to the bottom in a button, cooled and broken into pieces and transferred to the storage floor or bins at the blast furnace. It is then smelted in a blast furnace with limestone, with the addition of ferruginous or silicious fluxes as may be necessary, coke being used as fuel for melting and anthracite as a reducing agent. The resulting "cupola" copper is tapped into blocks which are melted in reverberatory furnaces and refined; while the waste slag, low in copper, flows continuously from the slag spouts into pots or into granulating devices for removal to the slag dump.

The Lake Superior smelting industry and its methods had its origin in practice derived from Wales, Welsh refiners having been employed in the works first established some fifty years ago in Detroit, Mich., for the treatment of the Lake copper. From that beginning there has been a gradual and constant improvement up to the present time in methods and the thoroughness with which

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Michigan Smelter at Coles Creek.

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the operations are conducted, and an increase in furnace capacity from seven or eight tons of copper per charge to more than one hundred tons; and careful and extended research has reduced the operation of refining from an empirical proceeding almost to an exact science controlled by laboratory tests; yet practically the same means are employed at the present day as at first, and the same general principles underlie the practice in the four smelting plants of the district although the methods employed at each differ radically.

The old time reverberatory furnace, with a capacity of seven or eight tons of copper—increased later to as much as twenty tons—had a hearth eleven feet wide by fourteen feet long, with a fire box four feet square. The sides and roof were laid up with clay fire brick and the hearth was put in with a highly refractory sand carefully baked in layers and seasoned with copper. Furnaces of this sort are still in use at three of the works. They are charged with mineral and mass in the afternoon, the fires are cleaned and built up and the melting continued during the night, slag being skimmed off at intervals until the gangue has been removed, and early in the morning the charge is ready for the refining operations. The first step, known as rabbling, is to oxidize such comparatively small quantities of impurities as occur, which come to the surface and are skimmed off. Fortunately, these impurities are more readily oxidized than the copper, sensitive though the latter is to the attack of oxygen; but unfortunately suboxide of copper is formed also, some of it being removed with the impurities in the slag, though most of it remains in solution in the molten charge. This then must be reduced back to metallic copper by the introduction of carbonaceous material, poles of hardwood being used for this purpose and buried deep into the copper. During these periods small quantities of slag formed by the oxides are skimmed off. The completion of the rabbling, or the extent to which the oxidation is carried, is indicated to an experienced observer by the character of the crystallization of the copper as exhibited in the fracture of a test button taken from time to time, the coarseness of crystallization which must be attained and the degree of oxidation to which the copper must be subjected depending on the nature of the material being refined and the proportion and kind of

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impurities present—the time required varying from two to six hours. By the expression "experienced observer" is meant not one who looks wise and mysterious when the button is shown to him—who appears to have some occult insight which guides him in judging the condition of the copper—but one who has learned by repeated comparisons of his observations with the final results ascertained by laboratory tests how to decide with some certainty that the rabbling has proceeded far enough. The usual test made when the copper is poled up or refined is for electrical conductivity, and while with cathode copper the button need be only finely granular to assure a conductivity of 100 per cent., to reach that figure with copper refined from the concentrated native mineral the button must be coarsely crystalline; while in refining the cupola copper recovered from the slag the button may have the same coarse crystalline fracture and still the conductivity not be over 70 per cent. The amount of impurities is by no means uniform from one time to another even for the same kind of copper, and no general rule could be deduced to govern all cases because it would be necessary in order to apply the rule to determine precisely the kind and amount of impurities contained in each furnace charge—a proceeding which, for obvious reasons, is entirely out of the question. The test button is commonly taken with a small ladle two inches in diameter and the button is three fourths of an inch thick. It is placed in a vise, nicked and broken across the middle so as to expose the fracture and the small hole just under the top. Rabbling was formerly accomplished by stirring the surface by pushing a rabble—a four-by-six-inch piece of iron attached at a right angle to an iron handle eight feet long—back and forth; this laborious operation has been entirely superseded by the use of compressed air either blown through one-inch iron pipes inserted into the copper or made to impinge on the surface.

The refiner in charge of the furnace having satisfied himself by repeated tests that the copper is sufficiently purified submits a button to the foreman for his judgment; and if this is favorable, the poling is started and carried on until the surface of a test button sets level and without any depression when cold, this being the end of the refining process and the metal being then ready to be poured into moulds by one of the several methods employed at the different works.

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The slag skimmed from the reverberatory furnaces during the melting and refining operations contains from 15 to 30 per cent. copper and may be either acid, that is running 40 per cent SiO_2 and 20 per cent. Fe_2O_3; or basic, running 25 per cent. SiO_2 and 45 per cent. FeO—the former coming from the amygdaloid mineral and the latter from the conglomerate. When these different kinds of mineral happen to belong to different mining companies the resulting slag is kept separate because of the irregularity in the copper contents in slag from mineral charges and from charges refined from cupola blocks and the impossibility of obtaining a representative sample for assay, thus necessitating the use of iron for fluxing the acid slag and silica for the basic slag in order to produce a freely running slag approximating 33 per cent. each of SiO_2 and FeO, and 20 per cent. CaO. The iron is found in the hematite mined within a hundred miles or in pyrites residue shipped from sulphuric acid plants, while the silica is obtained in the form of sandstone quarried nearby. Limestone is either brought from Lake Erie by boat or by rail from quarries near the east end of Lake Superior. The blast furnaces—or cupolas—were formerly round or elliptical in plan, but are now all rectangular, varying from forty to forty-eight inches in width at the tuyeres and from eight to twelve feet in length, the crucible being built of fire brick lining within cast iron plates and the shaft consisting of steel water jackets. The shafts are from seven to ten feet high. No forehearths are used as, in the absence of matte which would keep hot, they would soon freeze up; so that the separation of the slag and copper is effected in the crucible—the deeper the crucible, the better the separation. In former years these furnaces were run with a blast pressure of as much as sixteen ounces but experience has shown that from three to four ounces is as high a pressure as should be used, thus permitting the furnace to run slowly and allowing the greater reduction and better settling of the copper. By lowering the blast pressure and lessening the speed from ten tons an hour to three, the percentage of copper in the waste slag has been reduced to one fifth of what it was under the former conditions. The blast is supplied by a Roots or Baker blower. The reduced and metallic copper, containing small and varying percentages of iron, sulphur and arsenic, settles to the bottom of the

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crucible and is tapped out at intervals whenever enough has accumulated to make a block weighing, at different works, from 300 to 1,000 pounds, while the slag, floating on the copper, flows continuously through a spout which inclines upward to form a trap to prevent egress of the blast.

The Michigan Smelting Works, situated on the south shore of Portage Lake some three miles west of Houghton, are the latest built, having been erected in 1904, and contain features new to the Lake practice although common in the West and East. The general scheme of the reverberatory plant is a combination of separate melting and refining furnaces—although the latter are also utilized to a certain extent for melting. This plan, which was previously tentatively pursued elsewhere under adverse conditions, works out very satisfactorily when applied in the comprehensive manner in evidence there. The bulk of the mineral concentrates is melted in the melting furnaces proper, and the only time lost is the few minutes occupied in running the copper over to the adjoining refining furnace which sets at a lower level at one side with the bridge a little in advance of the front of the melting furnace. The mineral is received at the smelting works by rail in hopper bottom steel cars of 100,000 pounds capacity, weighed on a track scale, and unloaded into circular steel bins situated high on the hillside, from which it is drawn into small steel hopper cars propelled by electric locomotives to the furnace hoppers. As soon as the copper is tapped out each morning the furnace is charged and the melting proceeds almost without interruption, thus securing the greatest possible efficiency from the furnace and avoiding the heat losses and strains and the loss of time—which would be from twelve to fifteen hours—incident to the refining operation. This arrangement also permits the adaptation of the furnace design to the purpose intended, a broad shallow hearth being used in the melting furnace while a deep narrow hearth is used for refining. In order to take advantage of the time not occupied in refining, the secondary furnace is charged at the close of the refining operation with such a quantity of the richer grades of mineral as can be melted and made ready by the time the primary furnace is ready to tap, this giving a well-balanced scheme. Bituminous coal is supplied to the furnaces from bins by electric tram cars

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and discharged into hoppers through coarse gratings which hold back large lumps that would choke the hopper gates. In the melting furnaces the coal is dropped through the roof of the fire box, but at the refining furnace it is shovelled in through end doors.

The refining furnaces are equipped with Walker casting machines, which are now very largely used in copper refineries, consisting essentially of a circular horizontal cast iron frame with heavy movable brackets or arms projecting radially outward to serve as supports for the moulds. The machine is rotated about a central pivot by an electric motor controlled by an operator who sits in the center, who also operates the hydraulic power for tilting the ladle into which the copper flows as it is tapped from the refining furnace and from which it is poured by him into the moulds as they are brought successively in front of it. As the machine rotates the copper chills or "sets" and becomes solid—the diameter of the mould ring and the speed of rotation being designed with this end in view—just as the moulds reach guides which engage with the lugs fastened to the moulds and overturn them so that the copper ingot, bar or cake may drop into water for the final cooling upon a conveyor which removes it from the cooling tank and elevates it above the floor line and deposits it on a table for convenient transfer to steel platform cars of 30,000 pounds capacity running on a 36-inch gauge track and hauled by an electric locomotive to the scale and thence to the loading platform. More than 300,000 pounds of copper have been refined in a single charge in one furnace. The space in front of the refining furnaces is commanded by an overhead electric traveling crane with which poles, scrap, moulds and other material are expeditiously and economically handled. Indeed the use of electricity as a motive power has been very thoughtfully applied, and the absence of hand labor is very noticeable. The waste heat from the furnaces is passed through boilers in which sufficient steam is generated to furnish all the motive power, such as the electrical generator engines, pumps and air compressors.

The blast furnace is used for smelting mass copper as well as the reverberatory slag. This slag, skimmed into long narrow moulds, is broken up in a rock crusher and elevated into bins from which it is carried to the furnace in side-dump steel cars pushed

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by an electric locomotive, together with the fluxes and fuel. The tram track runs longitudinally through the furnace at the charging floor level and as the charging car passes over it the side gates are opened and the charge is distributed along the furnace. As the top of the furnace is normally cool there is no danger of overheating the rails or the cars, and when the heat rises, as in starting up and blowing down, the track is swung up to one side. The waste slag is granulated by dropping it into a stream of water flowing rapidly in front of the settler and is carried by the water through a launder to the slag dump in the lake.

Next in geographical order and in time of erection are the Quincy Smelting Works, built in 1898 on the north shore of Portage Lake, opposite Houghton. There is one main refinery building containing four separate reverberatory furnaces in each of which the mineral is melted and the resulting copper refined. The mineral comes from the stamp mill in 60,000 pound capacity steel hopper cars having separate compartments for the different grades, and is unloaded into storage bins from an overhead trestle. Small steel charging cars pushed by hand are used to transport the mineral from the bins to the furnaces where the cars are lifted by electric jib cranes and emptied into the hoppers above the furnaces. The furnaces are charged each afternoon, the melting and skimming goes on during the night and the copper is refined and dipped out the next morning. These furnaces have fixed boshes on which the moulds are placed, and contain water into which the copper drops when the moulds are overturned. The dipping is performed with ladles swung on trolleys traveling on I beams which run over the front door of the furnace and over the moulds. The copper is fished by hand from the water when cool and piled on hand trucks for weighing and removal to the shipping dock.

The blast furnace is charged with two-wheeled buggies from bins. The waste slag runs from the settler into a large slag car holding 3,000 pounds, mounted on four wheels. When this car is full it is run to an elevator and lifted about thirty feet above the ground level to the slag tramway, along which it is propelled by a Baldwin-Westinghouse locomotive to the slag dump, which forms a broad solid deposit on low ground.

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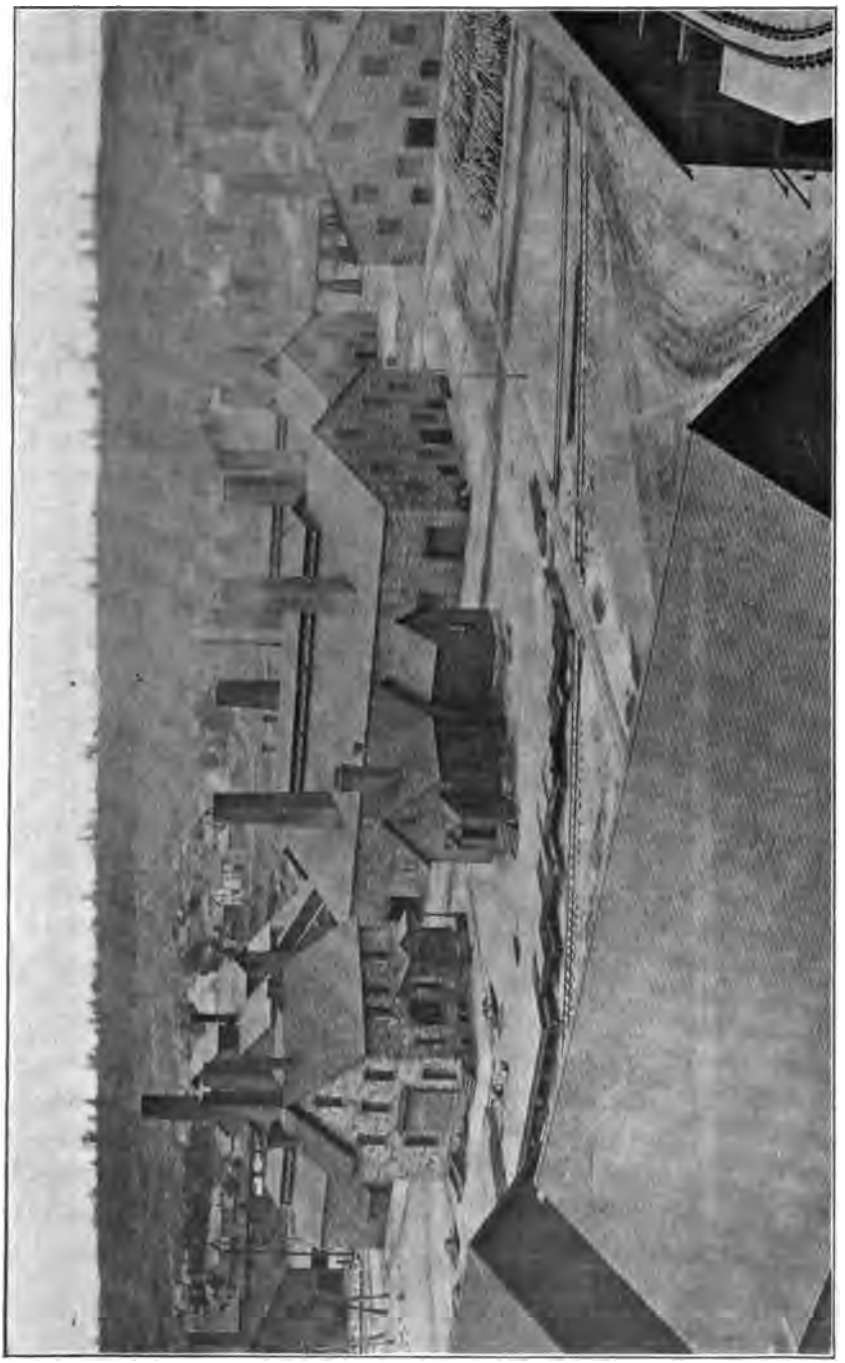

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There is a briquetting plant for making briquettes of the very fine slimes that form part of the concentrates. These are mixed with fine lime and pressed into form in a White briquette machine; then the briquettes are piled on tram cars and placed in a steel cylinder in which they are hermetically sealed and subjected to intense heat in an atmosphere of superheated steam. This puts them in condition suitable for smelting in the blast furnace without excessive loss in flue dust.

The Lake Superior Smelting Works at Dollar Bay, some three miles east of Houghton, on the north side of Portage Lake, were built in 1888. As originally planned there were two refinery buildings, each intended to contain four furnaces. After being remodelled twice the plant now comprises three large reverberatory furnaces and four small furnaces, the latter soon to be superseded by the fourth large furnace now under construction. Each furnace is used for both melting and refining. The large furnaces, seventeen feet wide by thirty feet long, are equipped with mechanical pouring devices and have a capacity of 200,000 pounds of refined copper to the charge. The casting machine consists of a series of cast-iron plates connected on each side by an endless chain which passes over sprocket wheels at each end, carrying copper moulds and bringing them successively in front of a hydraulically operated ladle kept constantly filled with copper flowing from the furnace. The ladle, in addition to the tilting motion, has a reciprocating motion alongside the table and is centered and carried along with the mould during the pouring by a lug which engages with one of the links of the chain and on being released at the end of the stroke is returned to the initial position by means of a hydraulic cylinder, thus permitting the continuous motion of the moulds without having to stop them at the ladle. The ladle is controlled by one operator seated opposite the tap hole. As the moulds pass down over the end wheels the solidified copper drops from them into the water of the cooling tank on to a moving conveyor which carries it along in the water until thoroughly cooled and then up to the loading table. From this it is lifted over to four-wheeled trucks that carry between 5,000 and 6,000 pounds, which are pulled by a gasolene motor truck to the scales, two in number, and thence to the dock or platform where it is piled in

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regular order ready for shipment. One of the scales has an ordinary beam which is read in the usual manner while the other has a type-registering beam which makes a printed record when the load is balanced, thus affording an absolutely reliable check on the reading at the first scale.

The mineral is brought to the works in steel hopper cars of 100,000 pounds capacity which are run up a trestle over the bins, these, however, being little used as the mineral is ordinarily dropped into hoppers leading to belt conveyors which carry it direct to the furnace hoppers—the cars being weighed both loaded and empty on a railroad track scale provided with a type-registering beam.

An electric overhead monorail crane traversing the works and yard serves to handle cupola blocks, mass copper, supplies and other material, delivers coal to the furnaces by means of a detachable shovel of the clam shell type provided for that purpose, which can be removed when not in use, and hauls the slag pots containing the reverberatory slag to the blast furnace charging floor.

Steam for running the power plant is obtained from vertical water tube boilers which utilize the waste heat from the furnaces. Each furnace also has a stack through which the gases may be diverted in case repairs to the boilers are necessary. Recording pyrometers and draft gauges making 24-hour graphical records indicate the working of the furnaces and guide the furnace men in their work.

At the blast furnace the slag, fuel and fluxes are shoveled into small steel tram cars which are pushed by hand to the charging doors and up-ended by a steam cylinder hoist so that the contents are slid into the furnace. The waste slag is granulated by falling into a stream of water and elevated by a link belt elevator into a storage tower from which it is hauled away by teams to the dumping ground. A recording blast gauge makes a continuous record of the blast pressure, which, together with the daily analyses of the waste slag, gives a check on the running of the furnace.

The Calumet & Hecla Smelting Works, built in 1885, are situated on the shore of Torch Lake, nine miles from Houghton and contain twelve reverberatory furnaces. The mineral at these works is stored for a time in bins to allow the water to drain out, after

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which it is drawn to the furnaces in two-wheeled buggies. These are lifted over the charging hoppers above the furnaces by electric cranes. At the blast furnace the slag, fluxes and fuel are lifted in the tram cars by an electric monorail crane and discharged into steel bins arranged around the sides of the building high enough above the charging floor to permit steel buggies to be loaded through short chutes; the buggies are then wheeled to the furnace doors and the contents dumped into the furnace. The waste slag escapes through a trapped spout into slag pots mounted on wheels from which it is dumped when chilled on the ground, broken and lifted by a derrick to railroad cars for transportation to the dumping ground.

At each of the works a well-equipped physical and chemical laboratory is maintained primarily for the purpose of determining the electrical conductivity and copper percentage—high electrical conductivity being an essential quality of copper intended for wire-making and an indication in other copper of its freedom from injurious impurities in the smallest proportions, while the copper assay shows the perfection or degree of failure to reach perfection of the refining operation. We may, in fact, regard the conductivity as the test of the rabbling and the copper assay as the test of the poling; and under certain conditions either result may be high or low without affecting the other. In addition to these fundamental tests, the laboratory has to analyze the blast furnace slag, both to guide in proportioning the charges so as to obtain the most easily fusible slag and to guard against too high a loss of copper in the waste slag. In addition regular assays are made of samples taken from the mineral to ascertain the moisture and copper content, of the fuel and fluxes, furnace sand, etc. Besides the ordinary routine work considerable research work is carried on for the purpose of bringing to light unknown factors which would influence refining and the quality of the copper.

The laboratories are all roomy, well ventilated, modern buildings, especially arranged for this work, provided with sample crushers and grinders, drill presses for drilling copper samples, lathes for turning bars for making tensile strength tests, and rolls and wire drawing apparatus. Three different types of apparatus for making electrical conductivity tests are in use—the Hoopes

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direct-reading bridge, the Willyoung and the Siemens, all giving identical results when referred to a common standard such as that at Washington. The first has the advantage of being quicker as it is devised so that the result in percentage is read directly from a scale when the current through the test wire is balanced against that through the standard; and it has the further and principal advantage that the test wire is compared directly with the standard instead of through resistance boxes, the values of which have to be determined by deduction from the conductivity of the latter. The copper assays are made by the electrolytic method except that for coarse copper the crucible fire assay is used. For rapid copper determinations induction coils are used, consisting of a spool some three inches in diameter inside and six inches high wound with insulated copper wire; an electric current passing through this causes the rotation of the electrolyte contained in a beaker set within the spool, thus permitting fast deposition on a platinum gauze cathode of copper free from sponginess or gases, the time required being two hours or less as compared with sixteen to eighteen hours for the regular method, while the results are equally accurate.

The refined copper product is cast into a variety of shapes such as ingots, weighing about twenty pounds each, used for making brass, bronze, bearing metal, etc.; ingot bars consisting of two or three ingots joined together endways in one piece for convenience in shipping, thus avoiding the use of the casks required for small ingots; wire bars about four inches square and four and a half feet long, weighing 225 to 250 pounds, for rolling and drawing into wire for electrical purposes; cakes, both square and round, varying in weight from 120 to 6,000 pounds and from 14 inches by 17 inches up to 56 inches square, for rolling into sheets intended for making copper boilers, bathtubs, vacuum pans, digesters, etc.; billets for manufacturing seamless drawn tubes; and when there is an appreciable amount of silver associated with copper, into anodes for subsequent electrolytic treatment to recover the silver.

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A NEW SYNTHESIS AND NEW OCCURRENCES OF COVELLITE.

By AUSTIN F. ROGERS.*

Covellite from Galena, Kansas.

In the Joplin district copper minerals are rare, chalcopyrite being the only one at all common. At the Big Coon mine on West Seventh Street in Galena, Kans., an insignificant mine now abandoned, the writer found covellite, chalcopyrite, cuprite, malachite, aurichalcite, linarite, caledonite, and chrysocolla. The covellite occurs replacing sphalerite and pseudomorphous after it, but is not directly associated with any other copper mineral. An analysis made by Mr. Edgar B. Hayes gave the results in column I.

	I.	II.	III.
Cu	18.80	67.10	66.45
Zn	43.68		
Fe	1.17		
S	31.37	32.90	33.55
SiO_2	4.47		
Total	99.49	100.00	100.00

Calculating the Zn as ZnS and the Fe as FeS and deducting their sum, which represents unchanged sphalerite, together with the SiO_2 and recalculating to 100 per cent., we have the figures in column II. Comparing this with the theoretical for covellite given in column III., we have proof that the mineral replacing sphalerite is covellite. It is blue-black in color, has a submetallic luster and a hardness of about 2. It seems probable that sulfate water with copper in solution has acted upon sphalerite, producing cupric sulfid according to the metathetic reaction:

$$ZnS + CuSO_4 = CuS + ZnSO_4.$$

A New Synthesis of Covellite.

Experiments made by the writer in 1899 showed that the reaction mentioned takes place on heating the proper constituents in a

* Ph.D. Columbia, 1902.

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sealed tube at a temperature of 150°–200° C., but quantitative proof was lacking.* Recently this experiment has been repeated in the writer's laboratory. A half of a gram of finely powdered sphalerite from Santander, Spain,† and a water solution of an equivalent amount of copper sulfate, plus a slight excess, were placed in a hard glass tube about a foot long. Carbon dioxid was introduced to prevent oxidation. The sealed glass tube with contents was heated in a bomb furnace on two successive days for seven hours each at a temperature of 150°–160° C. When taken from the furnace the solution was pale in color and instead of the sphalerite there was a dark substance with the typical blue-black color of covellite. To prove that this was covellite an analysis of it was made by Mr. Horace Mann, a geology student at Stanford University. The results are given in column I.

	I.	II.	III.
Cu	60.10	66.41	66.45
Zn	5.98		
Fe	0.74		
S	33.75	33.59	33.55
Total	100.75	100.00	100.00

Calculating Zn as ZnS, which represents the unchanged sphalerite, and the Fe as FeS, assuming it to be in this form, deducting these and recalculating to 100 per cent., we have the figures in column II. The theoretical percentages for covellite are given in column III.

This is proof that the material is cupric sulfid and so this is a new synthesis, or rather metathesis, of covellite. A microscopic examination of the material showed opaque fragments of the same shape as the original sphalerite cleavage flakes. There is replacement of the sphalerite but it is not complete, for on crushing the fragments a little of the sphalerite is visible with the microscope. A recalculation of the analysis gives 90.5 per cent. of covellite and 8.9 per cent. of sphalerite.

The sealed-tube method represents a general type of synthesis. Increased temperature and increased pressure make up for geologic time. In the Joplin district there is no evidence of the action of thermal waters, there being a total absence of hot-water minerals.

* University Geological Survey of Kansas Reports, VIII., p. 460, 1904.

† According to an analysis by Caldwell this contains Zn, 66.59; S, 33.60; Fe, 0.16. Dana, "System of Mineralogy," sixth edition, p. 61.

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Covellite from Calaveras Co., California.

The writer has collected specimens from the Poole mine at Nassau and at a prospect between Nassau and Copperopolis in Calaveras County in which covellite has been formed by the replacement of sphalerite. These localities are in the foot-hill copper belt just west of the Mother Lode. The ores occur as fahlbands in schists which are more or less sericitic. The chief mineral is pyrite with some chalcopyrite and sphalerite. The chalcopyrite no doubt furnishes the copper but the sphalerite is sometimes replaced by covellite, a soft bluish-black mineral which gives sulfur in the closed tube and reacts for copper. The sphalerite when not apparent in the hand specimen may be recognized by crushed fragments with the microscope. Qualitative tests also showed the presence of zinc. This material which is a low-grade copper ore is an example of secondary sulfid enrichment on a small scale.

Covellite from Yerington, Nevada.

I am indebted to Mr. W. C. Orem, of Salt Lake City, for a small specimen of covellite from the Nevada-Douglas mine at Yerington, Nev. This is a porous material of dark indigo-blue color and contains a little pyrite. An examination of the crushed mineral

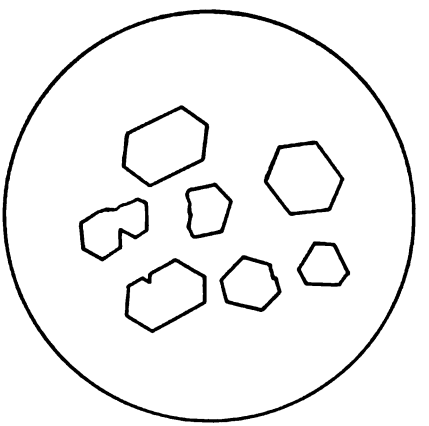

Fig. 1. Covellite, Yerington, Nev.

under the microscope reveals minute (.05 mm.) hexagonal crystals of tabular habit as represented in Fig. 1. Some of the smaller, and therefore thinner, crystals are green by transmitted light, while

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other crystals are opaque in the center and translucent on the edges. This character of covellite has been noted by von Bornemann at a locality in Sardinia and also by Weinschenk on artificial covellite. In this connection should be mentioned the fact that Schroeder van der Kalk obtained a green streak by rubbing covellite very thin with quartz. A colloidal cupric sulfid has been described which possesses green fluoresence. The fluoresence is perhaps due to the fact that cupric sulfid is blue by reflected light but green by transmitted light.

Covellite from the Rambler Mine, Wyoming.

This occurrence has been described by Read* and is simply mentioned here for the sake of completeness. Chalcopyrite and pyrite are associated with the covellite and the order of succession as made out by Read and verified by the writer is: (1) chalcopyrite, (2) covellite, (3) pyrite. As can be seen from the sketch (Fig. 2) of

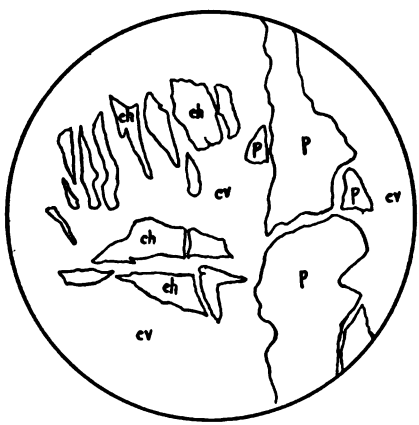

Fig. 2. cv. covellite. ch. chalcopyrite. p. pyrite.

a polished section the covellite has been formed from the chalcopyrite. Read gives the equation

$$CuFeS_2 + 2CuSO_4 = 2CuS + FeSO_4.$$

Covellite from Moonta, Australia.

Massive covellite with faint cleavage has chalcopyrite and a little pyrite scattered through it. The chalcopyrite is evidently the source of the covellite, as the sketch in Fig. 3 shows. A few

* *Eng. and Min. Jour.*, 79, p. 985, 1905.

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cubical and cubo-octahedral crystals of pyrite were formed later than the covellite as some of them occur in cavities.

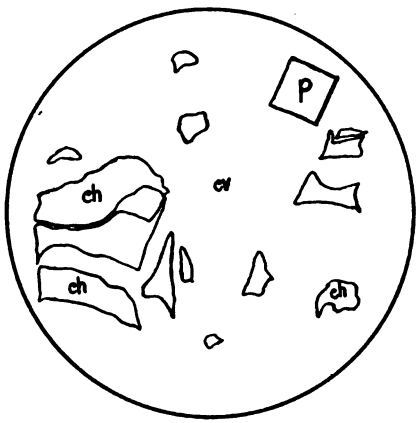

FIG. 3. cv. covellite. ch. chalcopyrite. p. pyrite.

COVELLITE FROM RUTH, NEVADA.

A specimen of high-grade copper ore from the Eureka ore-pit at Ruth, Nevada, was collected by Mr. Donald Steel and presented to the University. Covellite, chalcopyrite, and pyrite occur together. The chalcopyrite has, no doubt, furnished the covellite, but in this case the pyrite was probably formed before the covellite. The pyrite and chalcopyrite are intimately associated and the chalcopyrite often encloses the pyrite. It seems probable that a mixture of pyrite and chalcopyrite was originally present and of these only the chalcopyrite was acted upon by solutions producing covellite.

COVELLITE FROM THE CASHIN MINE, COLORADO.

A massive compact specimen collected at this locality by Professor G. F. Kay is quite different from ordinary covellite in appearance. The only associated minerals are quartz, barite and a stain of malachite. Nothing is known of this occurrence,* but from the granular appearance of the covellite one would suspect it to be a metasomatic replacement.

COVELLITE FROM THE BULLY HILL MINE, SHASTA COUNTY, CALIFORNIA.

A specimen collected at this locality by the writer shows covellite as a coating on chalcopyrite and evidently formed from it.

* Since writing this I have found that the Cashin mine was described by W. H. Emmons. Bull. U. S. Geol. Surv., No. 285, p. 125. 1905.

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The Occurrence and Origin of Covellite.

1. *With Chalcopyrite.*—Of forty-five localities for covellite given by Hintze in his "Handbuch der Mineralogie" and by the writer in the present paper in which associated minerals are mentioned, chalcopyrite occurs at twenty-four. At thirteen of these covellite is pseudomorphous after or coats chalcopyrite and, no doubt, owes its origin to the chalcopyrite.

There is no synthesis bearing directly on this origin. Read* gives the equation $CuFeS_2 + CuSO_4 = 2CuS + FeSO_4$. This type of chemical change is characteristic of the zone of secondary sulfid enrichment, sulfates from the oxidized zone acting on the unaltered sulfids below. Covellite, it will be remembered, is characteristic of this zone and is one of the very few minerals limited to it. The common association of pyrite with covellite may be explained by the subsequent reduction of the ferrous sulfate.

2. *With Chalcocite.*—At about ten localities chalcocite is associated with covellite, often in the intimate mixture known as digenite.† Knop‡ produced covellite by treating chalcocite with dilute HCl or H_2SO_4. Stokes§ produced covellite by treating chalcocite with ferric chlorid. The equation is $Cu_2S + 2FeCl_3 = CuS + CuCl_2 + 2FeCl_2$.

3. *With Bornite.*—Covellite occurs as an alteration product of bornite at the Matchless mine, Damaraland, Africa, and at Copper Mt., British Columbia, in a pegmatite. Catherinet,¶ who described the last occurrence, gives the equation $4Cu_3FeS_3 + 6O + 3H_2O = 12CuS + (Fe_2O_3)_2(H_2O)_3$.

4. *With Tetrahedrite.*—At Freudenstadt, Württemberg, tetrahedrite is replaced by covellite.

5. *With Enargite.*—At Mancayan, Philippine Islands, covellite is intimately associated with enargite, evidently as an alteration product.

6. *With Malachite.*—In the Schapbach Thal, Baden, covellite occurs as an alteration product of malachite. A synthesis by Doelter has a bearing on this occurrence. He obtained covellite by

* Loc. cit.

† Hintze, "Handbuch der Mineralogie," I^{er} Band, p. 524.

‡ *Neues Jahrb.*, p. 533, 1861.

§ Econ. Geol., 3, p. 22.

¶ *Eng. and Min. Journal*, 79, p. 985, 1905.

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heating malachite with a hydrogen sulfid solution in a sealed tube at a temperature of 80°–90° C.

7. *With Sphalerite.*—These occurrences have been mentioned in the early part of the paper and a synthesis having a direct bearing on the occurrence has been described. This manner of formation of covellite may possibly explain secondary enrichment of copper in other places than those mentioned by the writer.

8. *With Galena.*—At Gottesberg, Silesia; at Wheal Falmouth, Cornwall; at the Canton mine, Georgia, and in Polk County, Tenn., covellite occurs pseudomorphous after galena. At Cooper Hollow, Galena, Kansas, the writer* found covellite as an earthy bluish-black material in cavities of corroded galena. To explain this occurrence the equation $PbS + CuSO_4 = CuS + PbSO_4$ was given. Wells† proved that this reaction takes place when lead sulfid and copper sulfate solution are heated together in an atmosphere of hydrogen. This will explain the coating of covellite observed by Lacroix‡ on cleavages of galena associated with copper minerals at many French localities.

9. *With Anglesite.*—Covellite is intimately associated with anglesite at Siegen, Westphalia; at San Antonio di Gennamari, Sardinia; and in the Schapbach Thal, Baden. The equation given under 8 may explain this association, namely: $PbS + CuSO_4 = PbSO_4 + CuS$. If the reaction were complete there would be no associated galena.

10. *As a Volcanic Emanation.*—Covelli, after whom the mineral covellite was named, found covellite at Vesuvius as a volcanic emanation. The synthesis of Durocher explains this occurrence. He prepared cupric sulfid by the action of hydrogen sulfid upon vapors of cupric chlorid. The equation is $CuCl_2 + H_2S = CuS + 2HCl$.

STANFORD UNIVERSITY, CALIFORNIA,
March 11, 1911.

* University Geological Survey of Kansas Reports, VIII., p. 460, 1904.

† Econ. Geol., V., p. 4, 1910.

‡ "Mineralogie de la France," Tome 2, p. 474.

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UNWATERING EL COBRE MINES BY AIR LIFT.

By E. H. EMERSON.*

In 1910 an interesting experiment with the air lift was made at El Cobre Mines. The following are the results:

Conditions.

The water stood at the 200-foot level, held by two Cameron station pumps. The shaft was 800 feet deep, with one cage road, 4 × 8 feet, open to the 800-foot level. The other compartments were blocked with platforms. Water was acid, and contained up to 400 grs. Cu per cubic meter. A tunnel from the 50-foot level through the mountain allowed a reduction in the head.

Installation for Blowing.

Tests showed the shaft to be blocked by old timber and guides, so that a cone was made to run on an old cable guide, lowered in the center of the compartment with a two-ton weight. The wood pipe was built upon this and lowered as put together.

The wood pipe twisted around the center cable in a spiral, so that it had made a complete turn in 200 feet without doing the slightest harm. Three columns were built up on the cone and tied toegther with new 1½-inch rope, which, when wet, held them together very firmly. They were 750 feet long, each, and had no support, as the guide and lowering ropes were soon eaten off by the water.

With 600 feet under water and 150 feet above, the wood columns were nearly floating. After being water-soaked they became very heavy.

The wood pine was 10 inches inside diameter. Air pipe was 2½-inch iron pipe, which was thoroughly tarred and changed as it was eaten by the water.

The high limit of our compressors was 140 pounds pressure.

* Manager, El Cobre Mines, Cuba.

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Two pipes were used, the third being in reserve and operated in case of breaks in the others.

Results Obtained.

The out-flow was measured hourly by weir.

The head at the start was 150 feet, submergence of air pipe 50 per cent. Head of finish was 360 feet, with 39 per cent. submergence.

The water removed was 113,120,000 gallons. Cost 7.434 cents per 1,000 gallons.

The pipes averaged 950 gallons per minute, each, or 105 cubic feet of air for 100 gallons of water.

The lowest speed was with 360 feet head, 1,500 gallons per minute for the two pipes, or 160 cubic feet of free aid for 100 gallons of water.

Second Stage.

After reaching the 400-foot level, a 2,000-gallon pump was installed, and the discharge of the blow pipes turned into the pump sump.

Under no head, one 10-inch pipe threw 2,200 gallons per minute with air cut down. This was the pump limit on a burst of speed.

A special test was made in lowering the water for seven days. The results were as follows:

Seven Days Test Blowing and Pumping.

Feet lowered	57′ 6″
Feet lowered per day	9′ 2½″
Gallons per foot lowered	307,100
Total gallons pumped	17,659,980
Total gallons pumped per day	2,522,954
Total gallons pumped per minute	1,750
Head at start on blow pipe	21′ 6″
Head at finish on blow pipe	79′
Average head	50′ 3″
Submergence at start	179′ 89 per cent.
Submergence at finish	100′ 50 per cent.

Air used, 100 lbs. pressure.

2½-inch air pipe inside 10-inch wooden pipe, one pipe only used.

42.3 cubic feet of free air to 100 gallons of water.

Cost: 2.32 cents per 1,000 gallons.

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In this case the pipe was limited by the capacity of the pump. 3,500 to 4,000 cubic feet could be blown from the one pipe with 1,000 cubic feet of free air.

The actual inflow of the mine is 213 gallons per minute.

The cost per 1,000 gallons by blowing down 60 feet once in two months, and pumping the water to the surface, is 5.23 cents per 1,000 gallons.

The cost for the pump alone, working under her full capacity, is 2.91 cents per 1,000 gallons (with labor proportioned to the blowing), but handling only the inflow with pumps of small size the cost was 8.3 cents per 1,000 gallons.

There is, therefore, a saving of 3.02 cents per 1,000 gallons by the combination method of holding the water at the 400-foot level.

It is a real saving, since in flood time the full 2,000 gallons capacity is necessary, and, therefore, there is no unnecessary investment.

A great advantage is that an enormous sump is provided, which insures against drowning by the sudden floods.

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A SIMPLE ANALYSIS OF THE INDUCTION GENERATOR.

By J. H. MORECROFT.*

In one of the large electric power stations in New York City there have recently been installed five induction generators with a combined capacity of approximately 40,000 K.V.A. and it is quite likely that other large stations will in the future introduce generating units of this type. The induction generator has become commercially important only during the last year or two; because of this fact not much has been written on the subject, so that a simple analysis of the characteristics and operation of the machine may not be without value at this time.

That an induction motor driven above synchronous speed would, under certain conditions, act as a generator was perhaps first shown experimentally by E. Danielson, who described his experiment in the *Electrical World* for January 21, 1893. M. M. Leblanc was probably the first to surmise that such use of the induction motor was possible. In 1889 he furnished a mathematical solution of the problem of the induction motor and this analysis shows that, above synchronism, the motor will act as a generator, but so far as the writer can ascertain, he made no experimental investigation of this possibility. C. P. Steinmetz, in a paper delivered before the A. I. E. E. in 1897, referred to the use of the induction motor as a generator and pointed out some of its characteristics. A. Fr. McKissick reported to the A. I. E. E. (June, 1898) the results of some experimental work performed on a small induction generator. The paper described in detail the operating characteristics of the machine and the effect of varying load conditions, etc. M. Boucherot showed experimentally that it was possible to use condensers for the excitation of induction generators, instead of a synchronous alternator, as in the present usage. His experiments, which are extremely interesting, are

* Assistant professor of electrical engineering.

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described in the *Bulletin de la Société Internationale des Electriciens* for February, 1898.

This brief historical review will enable the reader to follow for himself the development of the induction generator. The following paper will contain the results obtained by the early experiments and will also be based on the results of some work recently carried on by the writer.

The principles involved in the action of the induction motor are undoubtedly appreciated by all those who may be interested enough to read this article but the interaction of the stator and rotor will be briefly reviewed here, the review to serve as a basis for the analysis of the induction generator.

The stator or stationary part, is wound with two or three sets of coils (interconnected or not), displaced from one another by equal angular distances. Such a winding, when furnished with proper polyphase currents, will produce a rotating magnetic field, as was first shown by M. M. Deprez in 1880 and afterwards applied by Tesle and Ferraris in the construction of the first induction motors during 1887–88.

The rotor consists of a laminated iron core in a periphery of which are inbedded the rotor conductors. These conductors may be merely copper bars, connected together at each end of the rotor by short circuiting rings (called the squirrel-cage winding) or the conductors may be connected in series with each other and in groups to produce what is called the phase wound rotor. In this type of winding the terminals of the coils are connected to slip rings mounted on the rotor shaft; the rings may be directly connected together, in which case the winding is practically short-circuited as in the squirrel cage rotor, or suitable resistances may be connected between slip rings, by which scheme the resistance of the rotor circuit may be varied as desired. As the two windings operate the same in so far as our analysis is concerned, no differentiation is here attempted.

The moving magnetic field produced by the stator utilizes the rotor core as part of its path and so in its rotation it must cut the rotor conductors. An E.M.F. is thereby produced which causes a current to circulate in the rotor windings. Now it is a fundamental principle that a conductor carrying current placed in a

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magnetic field will so react with the field that it is urged in a direction at right angles to itself and to the field. Furthermore the force on the conductor is in such a direction that the rotor tries to turn in the same direction as the field is rotating and, if free to move, the rotor will continue to accelerate until it has practically the same angular velocity as the field. The amount of slip (difference between speeds of rotor and field) depends upon how much torque the motor is required to exert; it will be less than 1 per cent. at no load and may be as high as 6 per cent. to 8 per cent. at full load.

The amount of E.M.F. generated in the rotor circuit depends directly on the slip, being zero at synchronous speed and a maximum value with the rotor stationary. The current in the rotor circuit also increases with the slip but not directly; the impedance of the rotor circuit increases with the slip because one of its components (*i. e.*, the reactance) depends directly upon the slip, being greatest at standstill and zero at synchronous speed. The resistance component of the impedance is nearly independent of slip.

The next step in analyzing the action of the induction motor is to remember that, from the standpoint of electrical circuits, it is exactly equivalent to a constant potential transformer of considerable leakage. As the validity of this assumption is important it will be given here in detail. It is at once evident that

$$I_2^2 = \frac{sE_2^2}{(r_2')^2 + \overline{sx_2}^2}, \qquad (1)$$

where

I_2 = current in any rotor circuit.

E_2 = E.M.F. induced in this circuit with rotor at standstill.

s = rotor slip in per cent.

r_2' = resistance of rotor circuit.

x_2 = reactance of rotor circuit when $s = 1$.

Now as E_2 is a fixed quantity, we have, for *any given value of* I_2 (as, *e. g.*, that required for full load)

$$\frac{I_2}{E_2} = k, \text{ a constant,}$$

so that

$$\frac{I_2^2}{E_2^2} = k^2.$$

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Therefore

$$\frac{s^2}{(r_2')^2 + sx_2^2} = k^2 \quad \text{or} \quad s^2 = \left(\frac{k^2}{1 - x^2k^2}\right)(r_2')^2,$$

which gives

$$s = ar_2' = a(r_2 + R) \qquad (2)$$

where a is a constant involving the value of I_2 for which the slip is required, r_2 = resistance of rotor winding and R = external resistance in series with rotor winding.

involving the value of I_2 for which the slip is required, r_2 = resistance of rotor winding and R = external resistance in series with rotor winding.

Now suppose that R is adjustable (a phase-wound rotor with adjustable external resistance), then it is evident that s may be made equal to 1 by properly selecting R. This means that eq. (1) which applies to a moving rotor, may be satisfied for any value whatever of I_2, with the rotor stationary, provided a proper value is selected for R.

But when the rotor is stationary it is evident that the rotor and stator windings are exactly equivalent to the secondary and primary of an ordinary transformer, the value of resistance in the secondary circuit of which must be suitably chosen to make $s = 1$ in eq. (2) for that value of a which results from the value of I_2 chosen in eq. (1). Therefore the current relations in a revolving induction motor are the same as those in a certain equivalent transformer whose load resistance is properly chosen. So that the equations and ideas involved in the operation of the ordinary transformer may be utilized in investigating the induction motor.

At synchronous speed there is no current in the rotor circuits of a properly designed polyphase induction motor, and the equivalent transformer would therefore be represented as operating with open secondary, as standstill the motor circuits correspond to a short circuited transformer. At any other speed the transformer secondary would be closed through a resistance the value of which would be calculated from eq. (2), putting $s = 1$ and calculating a from the value of I_2 at the speed selected.

The induction motor may now be represented by its equivalent electrical circuit, Fig. 1, where R represents the added external

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resistance to satisfy eq. (2). Now assume the stator and rotor turns are equal in number (1:1 transformer) then the electrically equivalent circuit may be represented by Fig. 2. The values of x_0

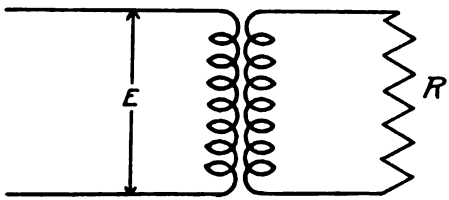

Fig. 1

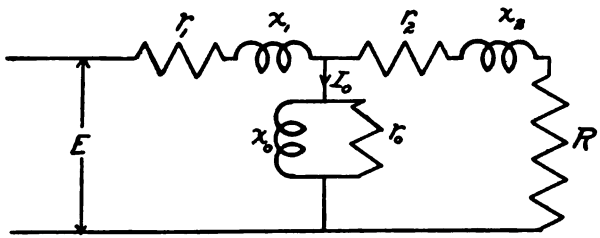

Fig 2

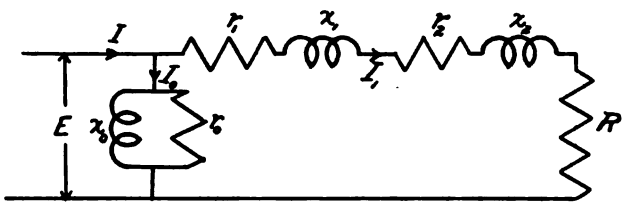

Fig 3

and r_0 are so chosen that the current I_0 is equal to the no-load current of the induction motor and the currents which flow through the paths x_0 and r_0 are equal respectively to the magnetizing and energy components of the no-load current, I_0. The letters r_1, r_2, x_1, x_2, represent the resistances and reactances of the primary and secondary respectively. Now as the drop of voltage through r_1 and x_1 is not very great even at full load, and as also I_0 is only a small part of full load current, the change which occurs in I_0 as the load varies from zero to rated value, may be neglected. Then the equivalent transformer circuit is depicted in Fig. 3. Of

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the quantities making up the impedance of this circuit R is the only variable. Referring to Fig. 3 it is seen that

$$I_1 = \frac{E}{\sqrt{(x_1 + x_2)^2 + (r_1 + r_2 + R)^2}},$$

which may be put

$$I_1 = \frac{E}{x_1 + x_2} \frac{x_1 + x_2}{\sqrt{(x_1 + x_2)^2 + (r_1 + r_2 + R)^2}}. \qquad (3)$$

Now this equation is in the form $x = A \cos \theta$ and is therefore the polar equation of a circle, x being the length of a chord, A the length of diameter and θ the angle between the two. So that the locus of current I_1, in Fig. 3, as R is varied, is a semicircle. Now the total current, I, is made up of I_0 and I_1. Hence the locus of I is also expressed by the semicircle of eq. (3) but the origin is shifted to an amount equal to I_0, both in magnitude and direction. The current which flows in an induction motor with various loads

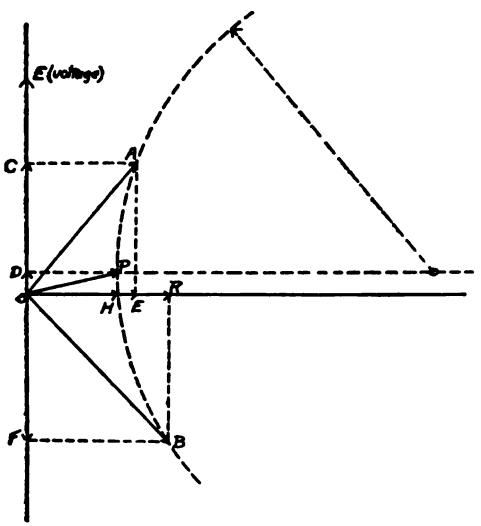

Fig 4

is therefore as represented in Fig. 4, where I, I_1, and I_0 have the significance previously given.

When R is very large (and I_1 correspondingly very small) I_1 will be nearly in phase with E. On the vector diagram this shows that the proper phase in which to represent the voltage is OE. As

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R decreases in value, I_1 gets farther out of phase with E until when $R = 0$, I_1 has such a position that

$$\tan \theta = \frac{r_1 + r_2}{x_1 + x_2}.$$

It will be noted that the net result of this transformer equivalence and application of circle diagram may be stated as follows: The current in the stator of an induction motor may be considered as made up of two components, I_0, the no-load current, and I_1 which is exactly equal to the value of current flowing in the rotor circuit. Really the phase of this current I_1 is opposite to that of the rotor current. This reversal of phase is involved in passing from the circuit depicted in Fig. 1 to that of Fig. 2. We may then conclude, whatever current flows in the rotor of an induction motor must have an exact equivalent, displaced in phase 180°, in the stator circuit and with this principle in mind we shall try to analyze the action of the induction generator.*

If the rotor of an induction motor is turning at a speed less than that of the magnetic field it is said to have position slip. The rotor conductors, cutting magnetic flux, generate what we will call a positive E.M.F. and there will flow in the rotor circuits a current which will be nearly in phase with this E.M.F. The lag angle of the current in the rotor circuit is given by

$$\theta_2 = \tan^{-1} \frac{sx_2}{r_2},$$

and for small values of slip this angle will be small. This rotor current must have its equivalent in the stator circuit and if the slip is positive, this stator current will be such that its product with the impressed E.M.F. represents *power input* to the machine; it is absorbing electrical power and giving off mechanical power, *i. e.*, running as a motor.

If the rotor is connected to some driving device (*e. g.*, another motor) and is driven at some speed higher than synchronous, it is at once evident that the E.M.F. generated in the rotor conductors

* It has not been the intention of the writer to rigidly prove the fact just stated but merely to review the process followed in the deduction. Some of the features involved, as *e. g.*, how a rotor current of frequency $= sf$ can have an exact equivalent in the stator circuit where the current frequency $= f$, have been omitted as not necessary for the purpose of the paper.

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will be in the opposite direction to what it previously had. Hence the direction of current in the rotor will be opposite and so must be its counterpart in the stator windings. This means, of course, that the current in the stator, due to the rotor currents, is in a direction opposite to what it had when the machine was operating as a motor. That is, the energy component of stator current is, for this condition, flowing in a direction opposite to the impressed E.M.F. Hence the machine must be *giving off* power and so is a generator. Such a machine is called an induction generator; as the frequency of current delivered is not equal to the rotational frequency of the machine it is sometimes called the asynchronous generator.

As the slip of an induction machine changes from positive to negative, it has just been shown that the energy component of the stator current changes phase 180° with respect to the impressed E.M.F. The wattless or magnetizing component of the stator remains in the same phase, however. With respect to the impressed E.M.F. of the motor this is a lagging current; reckoned from the phase of the motor C.E.M.F. it would therefore be a leading current so that referred to the phase of the generated E.M.F. of the asynchronous generator its magnetizing current is a leading current. For the induction generator to be operative, therefore, it must be connected to a line of certain frequency and some device in the line must be capable of drawing a leading current from the line, the magnitude of this leading current being the amount of magnetizing current required by the induction generator.

The magnitude of rotor current and hence of the stator current depends upon the amount of negative slip. The voltage of the generator depends upon the strength of its field, *i. e.*, upon the amount of magnetizing current furnished to it. These two characteristics are the distinguishing features of the induction generator.

An over-excited synchronous motor has both of the features necessary to make an induction generator operate, *i. e.*, definite frequency and a leading current. If therefore a synchronous motor with sufficient field excitation is connected to the terminals of an induction machine of higher rotational frequency than the motor, the induction machine will generate, supplying the amount

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of power to run the motor, and other load (as, *e. g.*, lamps) may be operated from the line.

In starting, the two machines may be electrically connected and both be brought up to a speed somewhere near the proper running speed of the induction machine, then the driving power may be taken from the synchronous machine and it will, if sufficiently excited, continue to run, drawing sufficient power from the generator to supply its stray power losses and taking also from the line a leading current which becomes the magnetizing current for the generator. As may be seen from this discussion the current relations in the induction generator follow the same laws as in the motor and as the circle diagram must hold good for both if it hold for the motor. In Fig. 4 the application of the circle diagram to the generator is indicated. The no-load current is shown at *OP;* for motor action some current as *OA* is required, having an energy component *OC* and magnetizing component *OE*. The current *OD* is required to furnish the no-load losses. If not loaded and not speeded up by a driver the machine operated at perhaps 99.8 per cent. synchronism and draws an energy current *OD* from the line. As the rotor is speeded up part of the energy required to supply machine losses is supplied by the driver mechanically and if the speed is increased to perhaps 100.3 per cent. synchronism, the stator current becomes *OH,* purely a magnetizing current and all of the machine losses are supplied mechanically. From 99.8 per cent. to 100.3 per cent. synchronism the machine gives neither mechanical nor electrical power; this is called the region of total loss. At some speed above 100.3 per cent. synchronism the energy component of stator current may go to some value as *OF,* in phase, opposite to *OC*. The total stator current *OB,* will be found on the same circular locus as *OA*.

An understanding of some of the generator characteristics may be obtained by considering what happens when a generator, excited by a floating synchronous motor, is loaded. Suppose that the speed of the generator is adjusted to give a speed of 60 cycles to the syncronous motor and the synchronous motor field is adjusted to give the induction generator a terminal voltage of 110 volts. If the generator rotor is held at constant speed and some load is connected to the generator terminals two effects will be

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observed. Both the voltage and frequency of the system will decrease. Even if the generator induced voltage remained constant the terminal voltage would fall with increase of load due to the increased IZ drop in the stator windings. But the induced voltage itself decreases with increase of load. The lag angle of the rotor current is $\tan^{-1} sx_2/r_2$, and due to the slowing down of the synchronous motor (explained below) s increases with load so that the rotor currents tend to demagnetize the stator more and more as the load is increased. This may be readily seen from Fig. 4. To give 110 volts terminal voltage, *e. g.*, at synchronous speed, requires a magnetizing current of OA. When the load on the generator is equal to OF, then the magnetizing current necessary to give 110 volts terminal E.M.F. is given by the vector OR. This current is made up of two parts; the original current OH and the demagnetizing component of the rotor current HR. Hence to maintain constant terminal voltage as the load increases the synchronous motor must be so excited that an any generator load the motor draws a leading current just equal to the wattless component of the stator current.

The synchronous motor slows down because it requires a certain amount of power to run itself and if the terminal voltage of the generator falls the C.E.M.F. of the synchronous motor must also decrease to permit the required current to flow. The slowing down of the synchronous motor may be explained from another standpoint. The stator current (load current) can only increase if the rotor current increases, as before noted. The rotor current can only increase if its slip increases and as the rotor is being driven at constant speed its slip can only increase by a slowing down of the field speed. Hence the synchronous motor must slow down sufficiently to make the difference between field speed and rotor speed such that the rotor currents generated produce in the stator just that current demanded by the load.

It will be noticed that this behavior of an induction generator excited by a floating synchronous motor is exactly similar to that of a shunt-wound D.C. generator with its brushes set in such a position that armature reaction demagnetizes the field. As load in such a machine increases the terminal E.M.F. drops because of the increased IR drop in the armature windings and because the

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induced voltage falls due to the weakening of the field by the armature reaction. If it is desired to maintain constant terminal voltage on such a machine the shunt field current must be increased sufficiently to overcome both of these effects. In the shunt D.C. machine this is accomplished by cutting out the field rheostat and in the case of the induction generator it is accomplished by increasing the field current of the synchronous motor. The decrease in frequency as the load increases can only be overcome by speeding up the rotor. With increase of load there must be corresponding increase in slip and if the rotor turns at constant speed the synchronous motor (by the speed of which the frequency of the power generated is determined) must slow down.

An interesting feature of the induction generator is the possibility of excitation by condensers. Although of not much com-

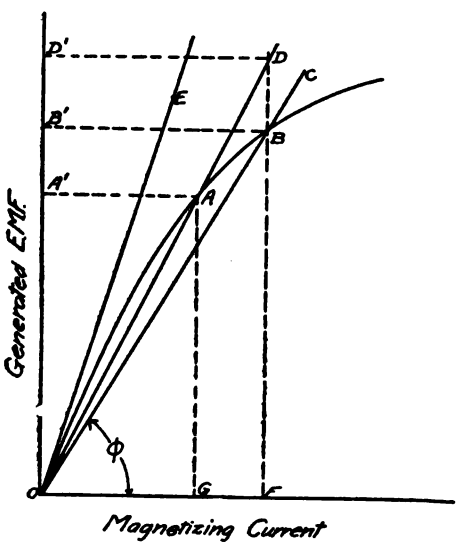

Fig 5

mercial importance at present an explanation of this method of operation will present the characteristics of the machine from a different standpoint and so will be considered here.

The magnetization curve of the generator will have the general form given in Fig. 5. The impedance of a condenser is $1/\omega c$ and the current in such a condenser is $E/\omega c$. The current is therefore

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a straight line function of the voltage and so may be represented by a straight graph using current and voltage for the abscissæ and ordinate. The equation of the graph will be $I = E \tan \phi$, where $\phi = \tan^{-1} 1/\omega c$. In Fig. 5 three such graphs are shown at *OC*, *OD* and *OE*, the graph *OC* being for a condenser of greater capacity than *OD*, etc. The magnetization curve of the generator is shown by the curve *OAB*. If the normal voltage of the generator = *OB'*, then the condenser whose graph is *OC* will just furnish the proper amount of leading current, *OF*, to generate the voltage *OB'*. A smaller condenser whose graph is *OD* requires a voltage of *OD'* to draw from the line a leading current of value = *OF*. But the generator when furnished with magnetizing current *OF*, can generate only voltage = *OB'*, which will cause to flow in the condenser a current equal to *OG*. Hence the condenser *D* would not serve to produce in the generator normal voltage. If the generator is operating with condenser *C* at normal voltage *OB'* and condenser *D* is substituted for *C*, the voltage of the generator will immediately fall to *OA'* at which voltage the magnetization curve and condenser graph intersect. With the condenser *E*, the generator could not operate because at no value of voltage does the condenser draw enough leading current to magnetize the generator to that same voltage; in other words, the magnetization curve and condenser graph do not cross. When once brought up to voltage the condenser *C* and *D* will serve to maintain the generator voltage but it is quite likely that if the generator is running idle and either of these condensers should be connected across the generator terminals it would not build up.

If the magnetization curve of Fig. 5 is much magnified and only its lower extremity considered we have Fig. 6. It is seen that the graphs *OC* and *OD* both have a region at the lower part of the magnetization curve where they lie above the magnetization curves of the generator. Of course, the generator could not "build up" through these regions because for a given magnetizing current the condensers require more voltage than the generator can give. To make the generator build up some condenser as *H* must be used, such that its graph *OH* does not intersect the magnetization curve in the region shown in Fig. 6, but in using such a condenser the generator is likely to build up to abnormal values of voltage, so

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as soon as the machine begins to generate the condenser must be cut down to its proper value.

When using condenser for excitation it is not at once evident how the frequency of the generated E.M.F. is determined.

The generator will have a certain inductance L and this combined with the capacity C makes a resonant circuit whose natural period of oscillation $= 2\pi\sqrt{LC}$ (neglecting the effect of resistance). Now it may be ascertained both mathematically and experimentally that the condenser used must be such that the natural period of the system is *less than the speed of rotation of the generator.* This only emphasizes the fundamental principles noted before, that the generator slip must be negative. The speed of the field rotation with condenser excitation must be brought lower than the rotor

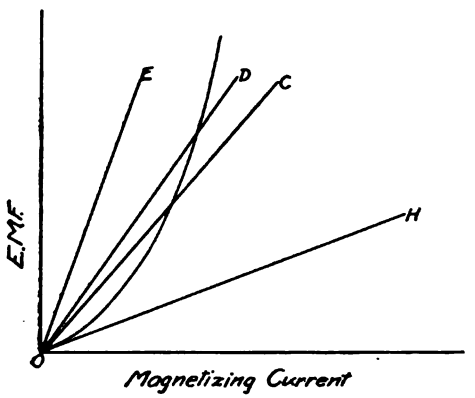

Fig 6

speed, and this is done by so increasing C that the natural period of the system has some value lower than rotor speed.

It has been previously noted that the external characteristic of an induction generator is the same as that of a shunt wound D.C. generator; if such a machine is short-circuited only a momentary rush of current occurs and the machine ceases to generate. It is because of this characteristic that the induction generator will probably be introduced in the large stations. It adds nothing to the short circuit risk of the plant. When a synchronous generator is short-circuited, many times the rated current will continue to flow until something in the circuit is burned out. A station may be equipped with 40,000 K.V.A. of synchronous generators and 40,-

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000 K.V.A. of a synchronous generator, the total capacity is 80,000 K.V.A. but the short circuit risk is only that of a 40,000 K.V.A. station. Of course there is the disadvantage that the synchronous machines operate with a power factor of about 92 per cent., as they must supply the magnetizing current for this induction machine. On the other hand the induction machine requires no D.C. exciter, as does the synchronous alternator.

The complete performance of a small single phase induction machine connected to a supply of constant E.M.F. and frequency, is shown by the curves of Fig. 7, showing the results of a laboratory

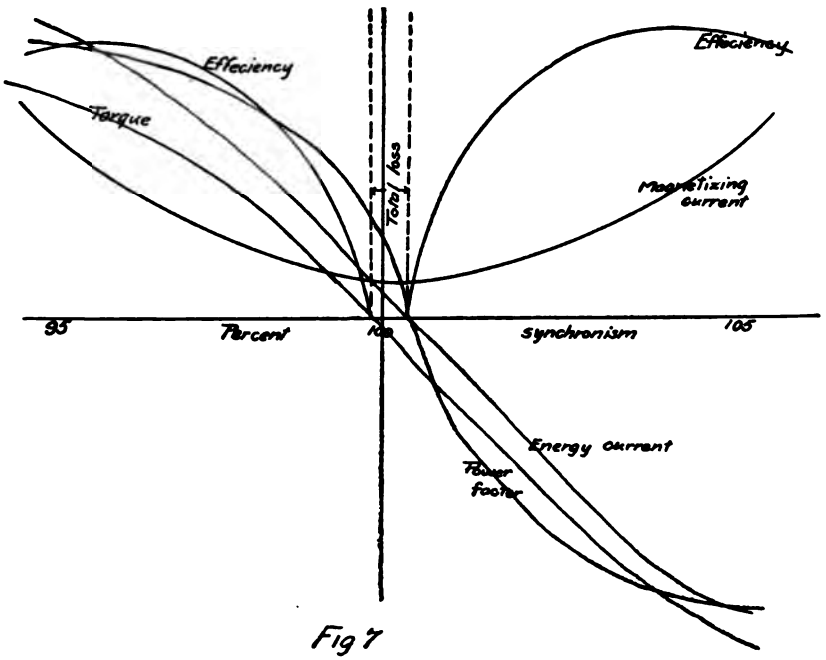

Fig 7

test through the complete range of motor and generator action. The maximum efficiency is slightly higher when running as generator but the range of slip in which the machine operates is considerably less when running as generator. If too much torque is exerted by the prime mover of the generator it will "pull through" and cease to generate. The machine tested would generate only between 100.4 per cent. and 104.8 per cent. synchronism. At 104.8 per cent. its maximum generator action occurred and any increase in the driv-

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ing torque resulted in the generator "letting go" of its load and a sudden rise in speed. This action is very similar to the stalling of an induction motor when too much load is put on it; the point of maximum torque is exceeded and the motor stops.

The construction of the induction generator makes it exceptionally suited for high speed as the rotor may have a squirrel cage winding which is very easily and firmly built. Practically no attention is required in so far as the electrical operation is concerned; no synchronizing is necessary. The induction generator is brought up to approximately the same electrical speed as the synchronous machine which is to supply its excitation and switched on to the bus bars. If then the driving torque is increased, it immediately begins to take load and requires no more adjustment. The voltage and frequency of the station output are determined entirely by the excitation and speed of the synchronous generators.

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ARCHITECTURE, ENGINEERING AND THE BUILDING LAW.*

DAVID H. RAY, C.E., Sc.D.†

The invitation to address you to-day on "Architecture, Engineering and the Building Law" comes rather opportunely, it seems to me, as it is my decennial year and I might present an inventory of ten years' experience. Just a decade ago Mr. Geo. B. Post, the architect, stood where I stand and I sat where you sit; and it is interesting to note that Mr. Post was invited by Professor Ware at my request, so that I am in a way a victim of my own idea.

But the time is limited, so I shall confine my inventory to one thought on architecture, one idea in engineering and a few words on the building law, and if I can deliver to you three sagacious observations from ten years' experience perhaps it is worth your attention and my endeavor.

First, as to architecture, I would draw your attention to what is given scant thought in the work-a-day rush of New York, namely the philosophy of architecture. This may sound preposterously academic, and even ridiculous as the topic of the best idea of the chief engineer of the Building Bureau. Yet I believe that all men have a philosophy of life, consciously or unconsciously, and that all architects have, or should have, a philosophy or theory of architecture.

The books, in a dim, uncertain way, indicate a theory of the development of architecture, indicating the influence of climate, geology, geography, and the social, religious and political conditions. In other words the environment determines the architecture. But the word environment smacks of science, of Darwinism, of evolution, and that is a horrid conception to bring into correlation

* Notes of an extemporaneous address at the School of Architecture of Columbia University, Feb. 21, 1911.

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with architecture—Queen of the Fine Arts; still that is what I purpose briefly to do, though I do not claim that it is anything new.

In the Metropolitan Museum of Art there is a model of the Parthenon and facing it a model of Notre Dame at Paris. Parthenos means "the virgin," Parthenon means temple of the virgin. "Notre Dame" is widely known as the synonym of "the Virgin." Indeed, the very same word Parthenos is used throughout the Greek Testament in reference to Mary. To the Athenians, the virgin daughter of Zeus was Notre Dame—Our Lady. In the original buildings in each case there is or was a statue of the virgin, and in the building and its vicinity, in each case, were erected the most sacred altars of the race. The same placard "A Shrine of the Virgin" might with truth and exactness be affixed to each model.

Both buildings are built in Europe, they are located about 1,500 miles apart and separated 1,500 years in time. The later is evidently at first glance the more complex. It is a higher evolution. The purpose of both buildings is identical. Each is a temple, a shrine, a house of prayer. Each is decorated and carved, expressing the aspiration and devotion of the surrounding community.

I commend to your meditation these two temples of the virgin facing each other in the Museum of Art, as indicating what a different architectural expression may be given to the idea, "A Shrine of the Virgin," by environment and fifteen centuries of evolution.

You remember that in Colonel Roosevelt's Romanes lectures he spoke on Biological Analogies in History. He spoke of the evolution of civilizations, of the fossil "three-toed horse," etc., but he did not mention the fossil remains of civilization, nor did he call attention to the evolution evident in these fossil remains. What I refer to is the ruins of cities—the fossils of civilization. This analogy is fully as good as any in his address and to an architect or engineer most striking.

The early civilizations were flimsy, as the early biological creatures were filmy and neither left any fossil. Then there are the tracks and trails in the dust of Nineveh, later in Egypt we find true fossil remains of civilization but the structure is rudimentary, the anatomy simple; there is no especial coördination. Greece exhibits fossil remains of a higher type, more unity, system and

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purpose; with Rome there is a further advance in coördination; the Gothic buildings are still more complex and so on, until finally there is born the building with a skeleton frame, the *vertebrate* class.

Now the point of my idea is just here. There has been evolved now a new class through or by a new principle, that is, a building with a frame or skeleton; it is a cohesive entity with special parts and circulation; one facetiously inclined might say: Yes, and it stands up on its hind legs too—at any rate it can and does. But the point is that it is just as foolish to dress this vertebrate skeleton construction in the fossil art of Egypt, Greece, Rome or Rouen as it would be for a biological vertebrate to parade about in the covering of the mollusca, echinoderma, arthropoda and other lower forms, or for a horse to wear the skin of a crocodile or of the dinosauros. The appropriate skin or covering of the vertebrate building—our new architecture in a narrow sense—will not come by pasting Greek and Gothic fossil facades on the skeleton.

My thought is that inevitably we are getting, and will soon have developed in the American environment an architecture expressive of this evolution in structure.

If we may imagine one of the giants of fable taking hold of the Parthenon by the roof, he would find the top stones in his hand—the thing did not cohere any more than a jelly fish coheres—but let him take hold of a modern American building by the top and it hangs together, it has a skeleton, it is of a different sort, and in the nature of things it will look different on the outside. Nor is it necessary that each part of the skeleton and each function should express itself on the exterior. In the human body the heart is on the left side and the liver is mostly on the right side, but one would never suspect this from the external architecture of the Appolo Belvidere or the Venus de Milo.

The appropriate outward dress will evolve itself through patient, sincere, serious endeavor and probably the use of reinforced concrete will be a considerable factor in the result. Nor can it be done at once, for an endeavor to make it out of hand will result but in a hideous Frankenstein monster. The Chicago firm of Adler and Sullivan came very close to success in this endeavor.

Do not use the fossil forms blindly, they are beautiful to be sure,

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so are the crustacea. Do not try to make a new architecture all your own out of your inner consciousness. Do not be carried away by drawings in the flat; think in the solid. Do not either "ornament the construction, or construct the ornament" but build a building, and remember that it has four sides and insides. So many seem to think that the architect's concern is only with the front on the street, and that if he hangs up a historical facade, all is well.

If he is not careful it may soon be supporting an enormous parasitic electric sign on its back, advertising the merits of Heatherbloom Chewing Gum. If there must be signs, let the architect meet the requirements of commerce frankly and arrange them with some propriety and taste, that his building may not become but a pedestal for an outrageously ugly advertisement and his pretensions to esthetic influence in the community a screaming farce.

Apropos of the large signs which are such a disfigurement and a menace to our city—I believe there is one now 70 by 90 feet on top of a building—it is only fair to say that present conditions are due to laxity in the past before the advent of the present Superintendent of Buildings, Mr. Rudolph P. Miller, Columbia '88. Mr. Miller is making a determined effort to better conditions in this respect and in fact in every respect, and I am proud to have a part in his admirable administration.

Another point that I would make is, that every structure should fit its environment. This is rarely given judicious attention, even in buildings of a public and monumental character. On a recent trip to New England two marked examples of this pressed themselves on my attention. The new Greek Boston Museum of Fine Arts is fine archæology, but it is as pretty as a fossil pearl shell in a mud hole; one feels that it ought to be in a museum under a glass case. The "Bennet Fountain" on the New Haven Green, a delicate white marble replica of the beautiful choragic monument of Lysicrates, in sunny Athens, is in about as appropriate an environment and use in the snow and murky atmosphere of New Haven as a Devonian fern tree would be as a monument to Captain Peary at the North Pole. I have studied the Greek literature for five years, I have the highest admiration for Greek civilization and art, but the architecture of the balmy Ægean is surely out of place and environment on our icy, "stern and rock bound coast."

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Now as to the one idea in engineering. My plea in architecture is for philosophy; in engineering I wish to lay the emphasis on common sense. The best engineering consists usually in doing the natural, straightforward thing. It is often said that "the highest art is to conceal art." This means "don't do stunts." Good architecture never crowds the factor of safety; neither does good engineering, nor does nature.

The "mechanics of engineering" is only three hundred years old. For ten times that period buildings have been constructed with very considerable satisfaction and success. The method was the method of common sense using a good factor of safety, and the buildings looked the better for it.

Graphic statistics appears to have been first set forth by Simon Stevinus, of Bruges, a military engineer of Prince Maurice of Orange in his "Hypomnemata Mathematica," *i. e.*, Mathematical Memoranda, in 1608. The principle of moments was not proven till Varignon gave the solution in "Nouvelle Mecanique" in 1687. There was no theory of strength of materials till 1638, and no theory of dynamics till about the same time. Both of these branches of engineering were first set forth by Galileo in his "Discorsi." It is obvious therefore that without these fundamentals, there could be no "mechanics of engineering" in the exact sense.

The point here is, that now having learned to estimate the factor of safety with some accuracy, if we put this into practice and work each material up to its maximum safe stress it will affect the architecture profoundly, and for the worse, judged by the customary esthetic standards. Nature has provided in the course of evolution substantial factors of safety, and structures will fit into nature and be more natural and more pleasing and satisfactory, if you do not crowd the factor of safety down to the minimum.

This is just where architects and engineers often "fall out" and fail to understand each other, and then both might to advantage go and listen to nature's teachings and return wiser and better men. The architect often makes a fetish of *fossils*, the engineer is sometimes carried away by the serviceable *fancies* of mechanics to the belief that he knows all about the possibilities of matter under the action of forces. As a matter of fact, engineering is

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not an exact science and can only be made to appear so by certain fundamental assumptions which are at best but serviceable hypotheses.

As Professor Mah says, in the final chapter of his book on Mechanics, "the science of mechanics does not comprise the foundations, no, nor even a part of the world but only an *aspect* of it." Mechanics as a most amusing "conceptual short hand"; its tricks and devices are most valuable. With all respect to its masters and without disloyalty to the profession of engineer my word of advice is do not stray too far from common sense and do not crowd the factor of safety.

Engineering is not the art of getting the most for a dollar, it is the art of serving mankind by mechanical science, and it should not be forgotten that there is a service of beauty and propriety as well as of miserly economy. It was the practice of a very successful engineer to spend fully half of the available time allowed for any piece of work, in thinking out the problem of the design, the "parti" and arrangement; and then to use his materials with the innate true gentlemanly consideration which was part of his fine character. They returned the courtesy by faithful support and never failed him. This course added to his fame as a good engineer, for no structure of his ever collapsed. I commend this practice to you.

Finally as to the building law; like many things it had a humble origin, in New York at least, and "there is a reason." Listen to the edict of appointment of the first superintendent, chief engineer and chief inspector of buildings on Manhattan Island, New York City:

"Whereas both by correct information and of our own knowledge we have remarked the disorderly practice of putting hogpens and privies in the public roads and streets, . . . three surveyors of Buildings,

Lubert Van Duicklagen,
Cornelius Van Tienhoven,
Paulus Leenderse

are appointed.

Done at Fort Amsterdam, 1647 July 25.

by Petrus Stuyvesant, Director-General of New Netherlands, Curacoa and the Islands of the Sea; Captain and Commander."

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Listen to the first building ordinances promulgated on Manhattan Island: "As a number of the houses are built of wood and covered with reeds no wooden or platted chimneys will be permitted on houses between the fort and the fresh water. Fort Amsterdam 1648." Not so very unreasonable, yet it was probably regarded as tyranny in 1650. Our ordinances governing building have grown, but not so rapidly as the town.

Listen to this from a deed of sale about 1680.

"A good house, wind and water tight, on a lot 50′ x 100′ on the graft at Broad and Beaver Sts., is sold this day by Abr. Rycken to Jan Ruthensen for $380.—land for $100.—house for $280." These premises within a stone's throw of the present financial center of the world were sold for two cents a square foot, somewhat over two hundred years ago. Now the land would readily bring ten thousand times that price. But the building regulations have not varied in anything like that ratio. The point here is that building regulations are necessary and are *alterable*. The ordinance in 1648 forbidding "hog-pens" and "wooden chimneys plastered with mortar" are no longer on the books.

The present building laws in general are, or were once as necessary and reasonable as the early examples that I have quoted; if they have outlived their usefulness, they should be annulled or revised. This cannot be done by the officials of the Building Bureau, whose function is executive, not legislative or judicial.

The building code should express the resultant of the stresses of the environment, not only by way of limitation but also by way of guidance, and in the nature of things it comes very close to doing so. The remedy is always with the majority, even though there may be indifference and inertia to be overcome. The building law should therefore be accepted not as gratuitous interference, but as a public necessity, as a service and an assistance to you in the practice of your future profession, because it defines what past experience in the opinion of the majority indicates as necessary and desirable.

Under this last topic I would emphasize therefore obedience and discipline. I have served five years in a mounted militia company and I think the militia affords an excellent and valuable training in obedience, discipline and command. There is also the advantage

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that if one offers this form of civic service he is exempt from jury duty, which often calls upon one most inopportunely in the practice of architecture and engineering.

In conclusion I would commend to your attention therefore philosophy in architecture, I would emphasize common sense in engineering, and I would bespeak obedience to the building law for the common good. I trust that my promise to offer three sagacious suggestions has now in your opinion been fulfilled, and I thank the faculty most cordially for the honor of the invitation to speak in the university to-day and I thank you for your kind attention.

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EFFICIENCY OF THE ARCHITECT'S CLIENT.*

By HENRY SNYDER KISSAM.†

Some three years ago, I addressed the students here on "System, in the Business Management of an Architect's Office." That lecture, which was published by THE SCHOOL OF MINES QUARTERLY in November, 1909, was in substance an outlining of business organization for conducting the office work, such as would admit of a peaceful, profitable practice.

The subject I have chosen to speak on for to-day is such as comes to attention and is disposed of, before reaching the stage of having use for the information given in my earlier lecture, as to organizing and conducting an established practice; in fact that earlier lecture, on System, might be regarded as Chapter II., and the address of to-day, on Efficiency, as being part of Chapter I., of Architectural Practice. Some day I hope to find time to present here Chapter III., "On Entanglements, or the Legal Questions and Decisions Arising out of Architectural Practice," and so conclude a brief on the founding a pleasureable professional practice.

This afternoon I shall discuss: Why and how an architect should be selected.

The character of this lecture is essentially that of a "campaign document," addressed to all who are immersed in the unfathomable depths of commercial materialism, its substance being largely gathered together from my letters to clients, or to building committees, urging on or clearing up matters of uncertainty in their minds. This statement will explain its ultra-didactic style and a certain lack of coherence.

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* An address, from the point of view of the practicing architect, to the students in the course on "The Theory of Professional Practice" in the School of Architecture at Columbia University, New York City, on March 14, 1911. Given under the auspices of the "Committee on Education" of "The Society of Columbia University Architects."

† 37 Liberty Street, New York. A graduate in the class of 1886.

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When the average merchant determines to build, he overlooks the fact that he is embarking on a venture he knows nothing about. Accustomed as he is to dealing in staple commodities, he generally goes about a building undertaking in much the same way as he would if buying merchandise, and views the building operation similarly throughout to its completion.

His most common, first misstep is to have failed to consider the matter of time as an element of efficiency; he waits until the last moment before acting on a building impulse; he is oblivious to the fact that ample time is indispensable in which to carefully study out the best solution of his problem. Such undue haste at the outset is more frequently the cause of dissatisfaction at the finish than are all other detriments taken together.

Unacquainted as he is with the merits, duties, limits and differences of the services of architects and of the services of architect-building-contractors, he cannot judge of value in them, so he "gives his order" to whomsoever impresses him with the best "selling talk" on "service" or on "price" or on both, and not infrequently such choice is his second misstep for he may find, at the end, that he had chosen one who neither interpreted his needs, nor safeguarded his interests.

While the preliminary studies of the plans and of the elevations are being made, it becomes necessary to choose the materials and methods to be employed in the construction. Such choice in matters of æsthetic, utilitarian, and technical learning, of far-reaching consequences to the owner, should be made the subject of discussion with the owner, for it is usually necessary to remove from his mind mistaken ideas and to implant sound reasons in place of them in order that he may not commit a misstep in these matters. Oftentimes the unreasonable creations that we see have been produced when the architect finds he is not to be permitted to plan or to execute the building simpler, truer, and straighter (with respect to distinction in material, construction, and design) than the man who pays for it, unless he continuously has the time and the will to educate the owner to higher standards. When he is able to be a tutor to his client he should produce a building better than his client, which the mind and habit of the client can grow up to.

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When the drawings have been finished and approved and the costs have been estimated, then the owner will have a chance to decide what kind of building contract he desires employed; in ignorance of the suitability of any of them he usually urges the one inducing the keenest competition or affording opportunity for shopping around for the lowest prices, and it is necessary to be able to guide him in this step safely for his own best interests.

Now in reaching this stage of the operation, there has been action in four important particulars: First, an owner selects an architect; second, an architect secures a client; third, they select the methods and materials for the operations; fourth, they select a kind of building contract.

It would be interesting to present a discussion of each of these actions in detail, but the limitations of space require me in this article to confine this paper solely to the first topic.

Now having indicated the class bias of the merchant, as influencing him to committing some of the initial mistakes mentioned, upon his engaging in building operations, let us consider the manufacturer. We may expect that he will be influenced by the same class bias towards committing the same initial mistakes as is the merchant; it will appear that not a few manufacturers are further from comprehension and into deeper misconception of the value and of the nature of the services of the architect and of the builder, than is the merchant.

When the average owner of a successful industrial concern, by extending its facilities, seeks to increase its efficiency, he generally thinks only in terms of square feet of floor area for so many new machines; that is his first misstep. He does not apprehend the problem of design as fixing two enduring causes of low efficiency: *viz.*, faults of structure, which fasten excessive fixed charges on capital; faults of arrangement of parts, which compel excessive operating costs. Low overhead charges, decreased cost of production, increased output, reduced selling price, and raised profits, are elements of high efficiency which are lost when wastefulness is structurally imposed.

A manufacturer needs such services as will insure him against erecting a structure that will be expensive, for its purpose, to build and to operate. Those services are performed by a specialist. To

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secure high efficiency in low fixed charges, of cost, insurance, and depreciation, and in low operating costs, of an economical arrangement of parts, for performing connected successive series of acts, it is necessary to retain a specialist, to design and to conduct the work. He only can insure the attainment of the most suitable results, economically, enduringly, and quickly. The specialist is a product of years of experience through which has been developed a specially trained intelligence with which to solve problems in design; such as, of handling raw materials, finished product and waste, illumination, employees' welfare, sanitation, fire hazard, heating and ventilating, within the limits of efficient operation, profitable financing and safe building.

It is a part of the services of a specialist to inform a manufacturer of the latest practices; to give him expert advice in all things; to make preliminary studies, to prepare complete drawings with full and clear specifications, to procure competitive bids on them from carefully selected builders; and to safeguard his interests, throughout construction, for honest, efficient and sufficient service, from the builders, in quality and in quantity, of labor and of materials, used in his building. He does not take contracts for building; the interests of an owner and a designer must be identical; the interests of an owner and a builder can never be idential.

Not a few manufacturers recognize the need of a specialist, only after expensive experience. Such an one has, commonly, fallen a victim to one of three mistakes: First, he has undertaken his project with only his mechanical engineer to advise him; second, he has engaged an inefficient architect; third, he has placed himself in the hands of an architect-contractor.

There are good reasons why these three courses are mistakes, which I will explain, but which may be summed up here in the general statement that no human effort is worth carrying on, if it is such that no special study may find betterment for it.

1. The mechanical engineer of a plant can know its needs in planning, but he cannot be a skillful originator, a trained economist, nor an experienced judge in the art of design, in the science of building, or in the business of management as applied to building

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operations. He can't know the best practices and the latest developments as a specialist.

2. The architect who has not specialized at industrial work, may furnish general architectural services of integrity in his relations to the builders, securing high efficiency in labor and materials, by preventing substitutions of inferior materials, scamp work, unsafe work, mistakes, needless delay, dishonest charges on the part of contractors, but he cannot furnish the expert skill of experience, insuring against erecting a plant that will be expensive for its purposes to build and to operate because of its unsuitable arrangement of parts, extravagent use of materials, excessive insurance rate, insufficient forethought for installing machinery, plant, etc.

It is not the cost of an architect's advice that counts—it's the cost of carrying it out and of then having to operate under its results; therefore it pays to look first into the worth of the advice rather than to have to look later into the expense of correcting the faults it resulted in. It costs as much to employ an architect of general practice as to employ an architect of special practice—usually more.

3. The building company or contractor (architect-contractor), that furnishes drawings and specifications in order to be the builder for the work, offers a seemingly beneficial arrangement for an owner, but which is in fact fundamentally prejudicial to the owner. By such a relationship the builder is made the judge of his own work, and as every builder is a buyer of both labor and materials as well as a seller of them, he will judge his own work by the standard of its profitableness to himself. He sells to the owner to profit himself, not to satisfy the owner. Its operation I cannot discuss in this paper.

Let us consider the results of these characteristic mistakes of manufacturers. Assume we have an owner with a special problem to be solved. He needs the services of an efficient architect, *i. e.*, one who is a specialist in a certain line of work. But he doesn't know it. He does not know what are the vital relationships, duties, benefits and defects of architects and of builders, towards himself or towards each other, in law and in morals. He does not even know that he needs to know such things.

1. He knows the practical operation of his plant. His engineer

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knows the operation of its mechanical equipment. He doesn't know that that is insufficient preparation for proceeding. It seems to them sufficient, and they go ahead together and plan the new work and let contracts for it. What they don't know about the undertaking they have plunged into devising, and conducting, is going to come to them in bitter experience, out of which the owner will lose his surplus and the engineer may lose his job.

2. Or the owner being a little wiser retains the services of a general architect, not knowing that one architect is as good as another only when neither is any good. If he finds out at an early stage that the architect can't meet his wants he dismisses him and goes on with his engineer—to destruction. If he doesn't find out until the end, the architect has a law suit for his fees and the owner has a building unsuited to his needs. In either case he condemns the whole profession of architecture.

3. Perhaps the owner may escape both collaboration with his engineer and action with an inefficient architect, but falls into the hands of an architect-contractor; one who furnishes drawings for the purpose of obtaining the building contract. Then he has avoided being tied up to insufficiency and to inefficiency by tieing up to inequity and iniquity. He may get an interpretation of his needs but he can't get impartiality or integrity in producing it and he will get a structure most expensive in upkeep, because the builder both selects and applies labor and materials, and in these his interests as a builder in serving self for self-profit are irreconcilable with serving the owner for the owner's profit.

The merchant or manufacturer to whom is presented for consideration these mistakes of ill-advised action, upon entering into building operations, may be expected to conclude that, with respect to attaining high efficiency for his operations, the employment of a competent architect has value to the owner.

It will now be of interest to discuss how an owner should seek to know his needs, in order that he may proceed about having them satisfied. Such a discussion leads us to the consideration of the philosophy of the relationship.

It would be well if the precept, taught by Socrates, were written large in the minds of men—Γνὼθι σεάντὸν (Gnothi seauton), "Know thyself"—which was written in front of the temple of the oracle

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at Delphi, for that is the beginning of all knowledge. Such knowledge, when possessed by an owner, would enable him to discern and to reveal his needs in intercourse between owner and architect, where now that communion, for the architect, often otherwise has to be too largely in the domain of psychological analysis in order to reach sociological conclusions. That we may not all have the time or the ability for such excursions into moral science, while pursuing the quest of a competence, results in the not uncommon occurrence that an architect fails to apprehend an owner's needs when the owner fails to comprehend them himself. Out of that condition there is soon to arise a tragedy overflowing with both friction and waste.

It is our most difficult task to discover those wants which an owner possesses but is not conscious of as also to discern rightly those wants called " felt wants " that he is conscious of. So therefore, it is no easy undertaking for us to become good architects, and not remarkable that an owner might find an inefficient one. Nor, on the other hand, as we will rarely find that an owner who comes to us will prove to be an efficient co-worker in unravelling the skein of his ideas or desires, need we be surprised to find that few owners are good clients. From this you will infer that an inefficient architect is not necessarily such per se, but may be rendered such to a particular client through not having achieved a coördination of ideas with the client.

Now, the most wasteful fault with a client, who has not catalogued his wants and classified them, is that he has no fixed ideals. He has not determined upon what he will hold fast to, as his unconditional or supreme wants. He is uncertain as to what to have, he covets too much, so he revises his directions unceasingly, thereby injecting into the labor of attaining the best solution of the problem, perplexity instead of coöperating for clarity. This vacillation and uncertainty causes the making and remaking of set after set of preliminary studies of the plans; it wastes the architect's time, the owner's time, and defers the time for execution of the problem. Such conditions are also wasteful of the architect's margin of profit for his services; even frequently imposing upon him a net loss through no inefficiency of his own.

Analogous to this condition is that in which an owner harbors a

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misconception in moral rectitude. He not infrequently views the service arising out of having engaged drawings to be made for him as one of submitting goods on approval without cost, overlooking the limitation to that commercial custom involved in that when the goods have been cut to order they are not returnable but must be paid for. Or he entertains the motion of imposing at the payment of his architect's fee a contingent relationship whereby the fee is to be based upon what he then has in mind to cut his cost to, and not upon what the builder's proposals showed would be the cost of obtaining what he had ordered should be provided for.

He does not know that such views, common to transactions in commercial life, where an exchange of commodities takes place, become dishonorable when applied to professional service where there are no longer material things but conceptions, opinions, and mental activities are dealt in.

That it is not the intent of an erring owner to depart from common honesty in his relationships with an architect is attested by the fact that the expression of this kind of moral obliquity is quite as commonly met with in church elders as in men of no religious profession. It is rather to be assumed that the whole sum of business experience in trade, precludes the possession of clear views of, or the ability to approach with clear reasoning to the needs of exercising the higher right conduct and character called for in the relationships that are not based on trade. His experience has not given him the ability to conceive of the cost and value of service. His limit is to perceive the value in commodities. They are concrete things, which he has learned in trade cost something to the one who sells them, a cost which he can estimate.

While there is a close resemblance in the nature of the service rendered by the architect, the physician, and the lawyer, all being based upon the selling of advice and involving trust by the client in the value of something he cannot himself measure, one client differs from another as to appreciating this resemblance and of conducting himself uniformly in his relationships with each of them.

Engagements for services are made by the public with physicians or with lawyers and they are paid, for what is asked of them, whether the wants have been stated clearly and fully or not, and

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without regard to whether benefited or not through what was asked.

The only difference between the services rendered by an architect or a civil engineer and those rendered by a lawyer or a physician is in the fact that the former professions are put to an expense in rendering a service, they having a productive expense in the way of draughtsmen and an overhead charge in the way of office rent and for salaries of other members of their organization whom they must employ even when they have no work. The other professions require no such coöperating force as a prerequisite to doing any business.

One wouldn't ask a lawyer to prepare a case for action, or ask several of them at once—and then reject all, or all but one, and pay nothing. One wouldn't ask a civil engineer to make a reconnaissance for a railroad and only pay him if it was decided to proceed to build the line. Yet it is not unusual to have analogous propositions arise in architectural practice, and a large part of an architect's time is consumed in educating his clientele in their obligations to him.

We can now consider how an owner should select the architect qualified to give him efficient service for whatever undertaking he contemplates.

A merchant applies "reliability tests" for judging the worth of goods he may buy. Why should he not apply such a practice to the operation of seeking to engage the services of an architect? It is one the application of which he is familiar with. His mind is accustomed to apply such thoughts. It is to the reliability of the service, without regard to price, he needs to inquire into.

Reputable architects follow the same standardized rates as their minimum charges; there may be some few whose charges are lower, but they are not cheaper. It is not the sum of the cost but the sum of the services that will or can be done he should measure. Cheapness and low cost are not the same thing. Things that cost little are not always cheap. In both buying goods and buying services, it is not the difference in price which determines whether one is to get satisfaction or disappointment from the purchase, it is the soundness of the judgment upon which the selection was based. The best is always the cheapest, in buying service.

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It is not always what he has done, but how well he has been trained to do a thing that gives value to an architect's services. The promise of fitness to secure for an owner tangible benefits and to accomplish savings in a building operation is indicated through the quality of education, experience, and character of the architect.

Buying sound building advice is buying insurance against inefficiency, dishonesty and disaster. It is hazardous not to have it. The cost is insignificant, while the insurance is reliable and complete assurance against dangers and owner is not qualified to detect or to avert.

The value of the architect's services is not measured by what it costs but by its results, *i. e.*, what it gives and what it saves. What it can give is determined by training, experience and integrity, to which his successes and his clients testify. What it saves in quality and in quantity is in amount many times the cost of the fee for service. It is not an uncommon experience in manufacturing buildings, that through a single fee to an architect of say three fifths of a cent ($0.006) per cubic foot of contents of the building, paid but once, many thousands of ineffectively disposed cubic feet are brought to a continuous earning ability of six cents ($0.06) per cubic foot per year, for a continuous sequence of years.

The "best architect" for some particular service is that architect who possesses integrity, has benefited by long and deep study in arts and in sciences, has had a broad general experience in practice, receives the commendation of those he has served, and in addition has studied the special problems arising out of that particular service. Above all, the employment of an architect, procuring excellence in design, and resulting in a structure of note, attracting the attention of all who pass it—securing also unexcelled facilities for efficient operation—gives to an owner an inexhaustible subject and object for advertising purposes, at one fixed cost, far excelling in value the enormous yearly expenditures of daily press advertising done by some merchants for publicity.

In considering the reliability for service that owner acts wisely who rejects the architect that accepts and adopts the structural design for reinforced concrete or for steel frame structures, which has been made by a contractor's employee, instead of having skill in his own office with which to produce such design.

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Accepting such substantial service from an interested party, other than a client, is immoral practice. It impairs the architect's standing with all who must know it, it impairs the value of his advice and services to his client, and it has been known to jeopardize the stability of the structure. A not uncommon result is that the contractor, who privately furnished the architect with a structural design, is the winner of the competition for its execution. Not necessarily through connivance of the architect, but through advantages gained unfairly with respect to the rights and privileges of competitors, constituting a travesty of justice.

There are some owners who frankly ask, what is the use of employing an architect, and to such it is not difficult to make plain the benefits to be gained. Of those others who don't ask and won't listen, only an expensive experience can enlighten them. The contractor makes a larger profit, the work possesses inferior quality, and the owner makes a bad bargain, when he seeks the cheapness of not engaging an architect. The money paid out for mistakes or deceits which advice could have saved is wasted. It can never be regained. Would it not be better business to have secured the advice that would have saved that loss when that advice also enables discerning further ways in which to cut down the expense and to increase the earnings of the structure, i. *e.*, to attain higher efficiency.

Every building operation, even such as is of small consequence, usually is, and always should be, conducted by some form of contract. The form in general use is that approved by the National Association of Builders, and by the American Institute of Architects, in which there are clauses calling for interpretation, observance, direction and judgment.

When an owner has not employed an architect, who is to decide these questions? Both the owner and the contractor are tainted with bias and can neither of them be expected to judge equitably. Usually the courts have to decide when an architect does not, while the owner is paying for expensive repairs and remodelling, in an effort to mitigate irremediable disappointments.

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FIRE PREVENTION AND FIRE PROTECTION FOR MANUFACTURING PLANTS.*

By F. M. GRISWOLD, G.I.†

Gentlemen: Let me assure you that I very highly appreciate the privilege of addressing an assemblage of this character, realizing that I have before me men of more than ordinary intelligence, whose study under capable instructors has prepared them to readily grasp the importance of fire prevention and fire protection as applied to manufacturing plants as an essential element in plant-design, and who are therefore mentally fitted to assimilate the technical and practical features embodied in that new branch of science known as "fire protection engineering," a knowledge of which is, in my opinion, a necessity in the proper rounding out of the technical equipment of these young men who are about to venture into the field of endeavor and accomplishment for which they have prepared themselves, and wherein the opportunity for reward and honors is limited only by the capacity of those who strive.

In view of the recent establishment of a department in your university covering the science of "fire protection," I fear that it may not be possible for me to tell you very much with which you are not somewhat familiar through the course of lectures which have already been delivered on the subject; however, in a matter of such broad scope and transcendent importance to the welfare of the individual and to the nation as a whole, iteration and reiteration is not only excusable, but is a necessity, born of the fearful toll which the people of this country are paying on account of the ravages of fire, which is yearly dissipating hundreds of millions dollars of the values vested in our created resources, for the conservation of which the public conscience needs to be awakened to a proper sense of the responsibility which rests upon each indi-

* Lecture delivered to fourth year class on manufacturing plant-design, Columbia University, May 9, 1911.

† Fire protection engineer, of the Home Insurance Company.

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vidual to personally endeavor to check this needless waste of both life and property, which is the result of selfish carelessness and indifference in relation to fire prevention measures.

I trust therefore that you will pardon me if I say a few words on the subject of the fire waste in this country, before giving attention to more specific matters relative to fire prevention and fire protection as applied to manufacturing plants. Assuming that each of those within reach of my voice is more or less familiar with the activities of the officials of the general government in relation to the conservation of the natural resources of our country, and that most of you are ready to commend and encourage this somewhat tardy effort to husband the bounties which nature has so kindly provided and which we have heretofore so prodigally wasted, let me ask, how many of you have taken seriously to heart the almost criminally inexcusable dissipation of the created resources, the actual invested wealth of this country, which for each year, and for many years past, has been absolutely eliminated from the assets of the nation, as a needless sacrifice to the demon fire?

Fortunately nature may, and frequently does, restore the waste of its values which man permits or instigates, but when man permits or instigates the waste of acquired wealth by fire, it means an absolute and irrevocable destruction of so much accumulated value; fire resolves the combustible into its original elements, which man cannot re-assemble into an entity as an evidence of his capacity for creating resources, nor can he count these intangible elements as part of his assets, hence, he who would retain that which has been accumulated, must be his own conservator and forfend the day of obliteration of his holdings by constant and unceasing effort to prevent fire, and supplement such effort by the provision of reliable and efficient fire protection.

In order to impress upon your minds the enormity of the fire waste in this country, with its accompanying sacrifice of human life and injury to person, let me epitomize the statistics of many years into an average one year period as follows:

During one year (1907) fire caused the death of 1,449 persons and the injury of 5,654, according to statistics gathered by the U. S. Geological Survey.

Each year since shows an increasing loss of life and record of

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injuries, the human sacrifice keeping pace with increasing fire waste.

Each year $250,000,000 of tangible value is wasted by fire.

Each minute of each day of the year sees $500 in value rising in flame and smoke leaving an ash-pile as its pyre.

Each year the fire loss equals $2.65 per capita of our 95,000,000 of population.

Each year this needless loss equals a tax of $13.00 per each family of five of our population.

Each year shows a record of 40 fires to each 10,000 of our population.

This, gentlemen, is a very brief statement; there are only five items in it, but it serves to brand the American people as the most prodigally wasteful of all civilized communities in relation to the conservation of their acquired resources; when we compare this shameful record with that of the nations of continental Europe in respect to fire waste which we find as collated in a report of the United States Geological Survey, covering statistics of six leading foreign countries including cities having population comparable with a like number of the leading cities of this country, it is discovered that only eight fires to each 10,000 population take place each year, and that the fire loss in the same period is but 33 cents per capita in the cities, and but 48 cents per capita as a general average over all, assuredly under these conditions the American people have no occasion to "point with pride" to their record.

In attempting to answer the natural inquiry as to the cause of the lower fire waste in foreign countries as compared with the experience with our own, it may be stated that under the centralized forms of government peculiar to these older countries, the individual is held to strict personal responsibility not only for fires occurring within his own property, but also for any damage to his neighbors on account of fire which is the result of violation of law, or in consequence of culpable carelessness, and may not recover payment for loss until after judicial investigation has proven him not at fault; the same conditions exist in relation to the tenant in whose occupancy fire may take place.

In addition to thus placing personal responsibility upon both the owner and the tenant, these governments formulate and enforce

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wise building laws and regulate the hazards of occupancy with such strictness that an evasion of either condition is almost unknown, while in this free country, every man assumes to be a power in himself, having little regard for the rights of his neighbors, and generally construing the term "liberty" into that of broad license to do as he pleases with his own, with the direful result evidenced in our yearly fire heap.

While it is perhaps beyond the bounds of hope to secure in this country as close official supervision of the individual as is had abroad, I confess to a desire for just that modicum of paternalism which would serve to take care of the fools, guide the ignorant, caution the reckless and severely punish the vicious when responsible for preventable fires. Certainly if we are to hope for reduction in our fire waste, it must be sought in making the individual realize his personal responsibility not only for the proper care of his own holdings, but also as an obligation to his neighbors and for the welfare of the nation in the matter of fire prevention.

It is a truism to state that communities are formed by the aggregation of individual units, but it seems wise to impress upon you a realization of the fact, that if the unit can be brought into appreciation of its obligation to prevent fires, the benefit to the community as a whole will soon become apparent, and herein, it appears to me, is opened to you a field of education of the unit both by precept and example, for the cultivation of which your technical knowledge so exactly fits you, and I trust you will appreciate their importance and grasp the opportunity whenever and wherever it may come to you so to do.

Passing now to the matters of fire prevention and fire protection, I am confronted with such a mass of essential detail in the proper consideration of each subject as to almost despair of being able to convey to you a proper conception of their importance within the time allotted for my remarks, but in general may say that *fire prevention* covers such a wide scope as to compel special consideration as to methods for each plant as it comes under observation, no two being so alike in their needs as to permit generalization relating to details, but as the very foundation of fire protection is based upon the completeness and efficiency of fire prevention, this latter phase of the question wil first be given

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attention, and in this relation it may be said that the most important and basic element in fire prevention is included in the term "shop management," or in more homely terms, "*good housekeeping,*" which is an essential in fire prevention in every plant, whatever the nature of its occupancy, the character of its building construction or the completeness of its fire protection.

Acceptable practice in "*good housekeeping*" demands strict compliance with the following prime essentials in fire prevention:

First, the enforcement of rules which will insure cleanliness throughout the plant as a matter of daily practice, not only as a means by which the possibility of fire may be avoided, but as of profit.

(*a*) Floor sweepings; greasy lunch papers, oily wiping waste, paint, rags and like material subject to spontaneous ignition, should be deposited in "Standard" safety cans suitable for their reception, the contents of which should be safely disposed of each night, preferably to be burned under the boiler.

(*b*) Workingmen's clothes and overalls, when not in use should be kept in ventilated metal closets or lockers not in contact with readily combustible material.

(*c*) Oily metal turnings or filings should not be permitted to accumulate on wooden floors or be held in combustible receptacles, nor should they be mixed with combustible materials.

(*d*) All combustible process waste and other refuse should be carefully disposed of by removal from the buildings at the close of each day's work, and be safely deposited in location not endangering the plant in case of ignition of such refuse.

(*e*) Time should be allotted to operatives for cleaning machinery and disposing of oily wiping waste, and for the removal of combustible waste material prior to hour of closing shop for the day.

(*f*) All volatile and inflammable fluids should be kept in and used from "Standard" safety cans; not in excess of one day's supply of such should be kept inside of building at any time, and all unused portions should be removed to a place of safety outside of the plant at the close of the day's work.

(*g*) Heating and lighting systems should be maintained in a safe manner and be kept in good condition; steam pipes should

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not be in contact with woodwork or other combustibles; hot-air pipes or other heat conveying or producing devices should be carefully arranged to prevent overheating or ignition of combustibles.

(*h*) Open lights or flame of any character should be maintained in such position as to avoid ignition of combustibles; gas brackets should always be of the rigid pattern, preventing swinging.

(*i*) Open lights or flame of any character should never be permitted for use in the presence of inflammable or volatile materials, or where inflammable dust is liable to be present; incandescent electric lights in such localities should be of the keyless socket pattern and enclosed in wire guards, with operating switch located in an apartment separated from the inflammables.

(*j*) The use of so-called "parlor matches" or their equivalent should be strictly prohibited in all parts of the plant. If matches must be used, only those lighting on the prepared surface of the containing box or receptable should be permitted.

(*k*) The use of the incandescent electric current for lighting is the safest means of illumination, when the equipment is installed in strict conformity with the "National Electric Code" and its integrity insured by proper supervision of the equipment.

(*l*) All specially hazardous and dangerous processes or devices which may serve to cause or promote fire should, where possible, be carefully segregated and properly separated from communication with the plant in general, and also receive special consideration in relation to fire extinguishing appliances.

(*m*) Watchman's service should be maintained at all times when the plant is not in operation, and the record of service be shown on such mechanical device as will not permit evasion of duty; records should be examined and checked over, filed and dated each day.

(*n*) Discipline should be enforced and system be maintained by holding shop foreman or floor boss strictly responsible for the maintenance of established conditions, a written report covering these matters to be filed with manager each day.

In order to be assured of the best results from the careful observance of these "good housekeeping" rules it is necessary to give consideration to the matter of building construction, as the measure of efficiency in both fire prevention and fire protection is

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largely affected by the character of the structure, it being evident that a fire-resistive building, having open and smooth interior surfaces, without concealed spaces and with the minimum of combustible material in construction and interior fittings, would call for a less elaborate system of fire protection and present smaller opportunity for dangerous accumulations than would be the case where, as in ordinary joist or light construction, the whole interior is readily combustible, hence, assuming to roughly outline the essentials which should have consideration in designing a factory plant, the following suggestions are presented:

(*a*) Wherever possible, fire resistive material should be used in construction, avoiding combustible floors, roofs and roof houses, interior trim and fittings.

(*b*) Avoid "pockets" or concealed spaces in floors and walls, which serve to collect and harbor shop refuse, waste, etc.

(*c*) Separation of areas into the smallest units practicable for the use to which they are to be assigned; divisions between sections to be formed by standard fire-walls having only such openings as may not be avoided, each of which should be protected by "Standard" automatic fire doors on each side of the separating fire-wall.

(*d*) All openings in side walls at the angles of adjoining sections and toward all exposing structures or accumulations of combustibles should be protected with "Standard" fire shutters, or, where the exposure is not serious, by wired glass in "Standard" metal frames; the absence of proper protection at such openings facilitates the lateral spread of flame and thus promotes conflagration.

(*e*) Vertical openings through floors should be avoided, as they form channels for spread of flame from floor to floor with almost incredible rapidity, and present one of the most dangerous features to be found in modern construction from a fire prevention viewpoint.

(*f*) Elevators, stairways, belt and rope drive raceways, should be separated and enclosed in brick or fire resistive shafts, having "Standard" fireproof automatic doors at all openings to plant.

(*g*) Fire escapes should be constructed with fireproof stairways, enclosed in brick or five resistive shafts rising above roof, with

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outside balconies having doors swinging outward from building and inward from balcony to stairway escape; no openings from shaft to the building to be permitted.

While under the present advanced state of the art of building construction there would seem to be no valid objection to demanding conformity to the suggestions above laid down in relation to new plants; it is probable that there will come under your supervision changes in the design of plants already established and located in buildings of less desirable construction, and in such cases it will become necessary to so change the conditions affecting the fire hazard and "good housekeeping" as to bring them as nearly as possible to this higher standard, and this may frequently be accomplished at moderate expense. Reference to the National Board of Fire Underwriters' pamphlet covering "Uniform Requirements" relating to building construction, will doubtless prove of value in such instances.

Again treating of building construction in its broad sense, it may be well to attempt a short classification of various types of buildings in relation to their adaptability to best promote success in the efforts toward fire prevention and fire protection herein suggested:

First, may be considered standard fire resistive or the so-called fireproof construction which presents no easily ignitable surfaces, and should when properly constructed show no avoidable features which might serve to obstruct the fire protective devices.

The higher first cost of this class of construction is thoroughly justified through the fact that its term of life is practically unlimited, and the probable depreciation somewhere about one ninth of one per cent. per annum.

The question as to which of two classes of fire resistive construction is the better lies between the steel frame skeleton enclosed with fireproof tile and that which is enclosed with reinforced concrete. In my judgment, there is little to be said in favor on one over the other except from the fact that in reinforced concrete construction, it is an evidenced necessity to have expert control from the selection of the cement through all the processes until the concrete is set in its moulds, and this result is

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not easily obtainable with the ordinary class of employees available in construction.

Second, "Mill" or slow burning construction carries with it many features which are desirable in relation to the ready extinguishment of fire from the fact that while all of its interior construction may be combustible, the heavy plank floors and their supporting timbers offer no hidden spaces, nor are they readily attacked by flame. The cost of this class of construction over that of ordinary construction is justifiable by the fact that its lease of life is superior, and the cost of its up-keep is very small.

Third, ordinary or so-called light joist construction is particularly objectionable from the fact that it presents the largest exposed area subject to ready ignition, and offers the most difficult problems in relation to fire prevention and fire protection. Its low cost is very materially offset by the expensive up-keep and the high rate of depreciation to which it is subject ranging from 4 to 4½ per cent. per annum. It is, therefore, apparent that under the present condition of advanced ideas in building construction, the designer of new manufacturing plants should strenuously avoid the consideration of the ordinary or light joisted type of construction.

At this point it seems wise to make reference to the National Board Building Code which has been prepared under the advice and council of eminent architects and engineers throughout the country, and presents information which will prove of large value to all who may seek to secure the result of a concerted effort to regulate the construction of buildings throughout the country which will best serve to reduce the fire hazard.

In attempting to specifically designate the nature or character of fire extinguishing appliances best fitted for use in any manufacturing plant, we are confronted with conditions in relation to the character of building construction and the nature of its occupancy which may serve to influence, if not to indicate, the selection of the class of apparatus which will presumably best control the given situation. While the choice of appliances for the purpose covers a wide range, measuring in efficiency from the humble cask and pail of water up to and including chemical extinguishers, standpipes and hose, fire pumps and hydrants under private control and water supply, and to be further extended to the full paid and

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organized fire department under municipal control, but as the efficiency of each of these devices and methods depends on the unreliable factor of the human element to be on hand and to intelligently handle them in emergency, experience has taught us that only when in the hands of skilled firemen may there be hope of satisfactory results in the use of these devices, and while cheerfully admitting their value as auxiliary means of fire extinction, I desire to call your attention to a method or system of fire extinguishment which is neither controlled or influenced by the uncertainty of the human element, a device and system which is always and automatically prepared to put out a fire at the point and at the time of its occurrence, responding instantly to the extinguishment of the flame, the heat from which sets it in operation, and which is suitable for effective service in practically any class of structure and under any condition of possible fire hazard.

This device is known as *the automatic sprinkler,* and it has reached its present position of dependability as a fire extinguishing appliance through long years of study and experiment, the records showing that as far back as the year 1809 and again in the year 1812, patents were granted in England to Sir William Congreve, M.P., for "an apparatus for extinguishing fires which shall be called into action by the fire itself at its first breaking out, and which shall be brought to bear upon the precise part where the flames exist"; while the automatic operation of this ancient device depended upon the burning of a cord to release a weighted valve which was normally closed, the wording of the claim is nearly broad enough to cover the more perfected device of the present day equipment, and had this far-sighted conception been supplemented by half as much energy in commercially forwarding the introduction and use of the device at that time as has been the case with similar apparatus during the past thirty years in this country, the saving of human lives and the preservation of property values from the ravages of fire incident to its use would have established a record astounding in its proportions, and highly creditable to progressive civilization.

While the desire to secure means of automatically controlling the spread of fire prompted many inventive minds to undertake the task after the death of the Congreve conception, nothing of public

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interest seems to have occurred until about 1852, when a system of perforated iron pipes was introduced for fire protection in some eastern cotton mills, the operation of which depended upon the opening of a valve at the outbreak of a fire, thus calling in the human element with all of its uncertainties as the actuating power, and while this class of extinguisher did some good service between the date of its inception up to about the year 1875, the damage from the widespread distribution of water beyond the exact seat of the flame proved to be as disastrous as that caused by the fire itself, and the perforated pipe was replaced by a series of individual sprinkler heads each having a cap attached to its extremity and held in place by a fusible metal solder of such composition as to melt at a fixed temperature, thus releasing the cap and permitting the flow of the water; this was the invention of Henry S. Parmelee, a noted engineer, of New Haven, Conn., and the first record of a fire being extinguished by this device dates back to February 12, 1877, when the property of the American Linen Mills, at Fall River, Mass., was saved from destruction through the operation of the Parmelee sprinkler head.

Following this demonstration of the efficiency of the Parmelee automatic sprinkler as a fire extinguisher, the use of the device gradually extended, and the experience gained in the operation of same at a number of fires demonstrated that they were somewhat slow in action, owing to the fact that the fusible cap was in contact with the water of the system at all times, thus necessitating longer exposure to the action of heat to insure melting of the alloy and retarding the operation of the device; realizing the seriousness of this defect in the water-joint head, and using the Parmelee idea as a base, many attempts were made to produce a head or valve in which the fusible solder would be removed from water contact and yet permit the use of a seal to the outlet which would be water tight through the mechanical device of struts which would hold it in place until released by melting of the solder joint; many of these devices proved to be impracticable and unreliable in service and have passed into oblivion, but there still remains a number of these devices which have stood the test of time and have been perfected to such a degree of reliability and efficiency as to have become recognized standards, and of these it accords me

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great pleasure to be able, through the courtesy of the several manufacturers, to present for your examination and study a full set of each of the various types of automatic sprinklers which are now accepted as satisfactory under the rules and regulations of the National Board of Fire Underwriters, and for the presence of which, when properly installed, large rate concessions are usually granted.

In explaining the nature and functions of the automatic sprinkler, it may be well to say that while in the exhibit before you each of the various types shows differences in general design and in the location of the fusible joint, as well as in its form, the principle upon which each of these devices operates is identical, in that by the melting of the soldered joint which holds together the several parts of the strut or lever supporting the seal to the outlet of the valve or head, this supporting device falls apart, and the pressure of water forces the seal out of its place, and striking the distributing plate the water is sprayed over a floor space the diameter of which is proportionate to the height of the sprinkler head above the floor, and its volume is controlled by the head in feet or pressure in pounds at which the water is supplied at the outlet of the device, ranging from 12 gallons per minute at 5 pounds pressure up to 58 gallons per minute at 100 pounds pressure, through the standard open waterway of one half inch diameter common to each of these devices.

In each case the composition of the fusible alloy used is standardized to insure its melting and the separation of the parts which hold the water seal in place at fixed temperatures suited to varying conditions; in this exhibit, the head without color other than that of its composition, indicates the normal or "regular" fusible joint, which is intended to release at a temperature of approximately 160° Fahrenheit, the variations from this basis being indicated by the coloring of the sprinkler head, the white coated sprinkler indicating a melting or releasing temperature of 212, blue that of 286 and red that of 360° Fahrenheit respectively, thus enabling the inspector to decide on sight the appropriateness of the equipment in specially heated localities; while the head, nearly black in color, is specially prepared with a composition called "coro-proof" which is intended to prevent corrosion of the parts

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due to the presence of acids or other corrosive elements in the factory, the melting point of this composition being such that its use is practically restricted to locations in which the temperature does not largely exceed that of 160° provided for the "regular" fusible joint.

In each of these devices the composition of the metal forming its body and other exposed parts is of such nature as to prevent oxidation, while the material composing the water seal of the valve is of various composition, but in each case, likewise non-corrodible, and at the same time of such character as to prevent adherence at its seat under the influence of continued pressure.

In addition to these devices which are designed to operate automatically at a given rise of temperature caused by fire inside of a structure, we also have here specimens of approved devices for use in protecting structures from attack by flames originating in other and exposing structures by the delivery of a volume of water against the outside walls, upon the cornice and over the fronts of the windows of the building to be protected, thus forming a water-curtain as a stop to the entrance of flame from the burning exposure. These devices are known as "open sprinklers," and having no seal at the valve outlet, they are not automatic in action, but depend upon the human element to render them effective by the operation of opening a valve at the base of the riser through which water is to be conveyed to the distributer, but even with this disadvantage, the open sprinkler has so often proven its worth as a reliable fire-stop as to warrant its introduction as a means of protection against exposure fires whenever it is possible to secure installation of such an equipment.

It may also be noted that the only difference between the open sprinkler used for wall and window protection, and that designed for application to cornices, lies in the shape of the water distributing table or plate, the design in each instance being that which has given the best results in actual practice at fires.

In concluding this description and exhibit of approved automatic sprinklers, it is well to relieve the minds of those who fear the happening of large water damage through the leakage of these devices under normal conditions of temperature, or from excess pressure, it may be stated that the minimum pressure at which a

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leak is excusable, is fixed at 250 pounds to the square inch, and the temperature at which the fusible link may melt is held at 160° in order to secure compliance with the National Board rules before approval of the device, and the record shows so few failures in this respect as to place the matter outside of serious consideration as an objection to the introduction of the automatic sprinkler as a means of certain and reliable protection.

Trusting that I may have made plain to you the details of construction and the intended operation of the automatic sprinkler, permit me to present a short statement of what, in actual experience, this very valuable extinguishing device has accomplished in putting out fires in practically all classes of structures, with almost any conceivable hazard of occupancy, and in all parts of the world since the device has reached that stage of perfection which has enabled its use to make the record:

From figures compiled by the General Fire Extinguisher Company it is shown that before the more general introduction of automatic sprinklers in factories, the average cost per fire was $7,361, while under automatic sprinkler protection the average cost per fire in 13,476 cases covered by their records, amounted to but $277.26 each.

Supplementing this, the record of Sprinklered risks fires compiled by the National Fire Protection Association, covering a period of some 14 years, shows that of 10,171 fires listed, 64.25 per cent. were entirely extinguished by the action of the sprinklers; that 30.59 per cent. were held in check to such an extent as to permit extinguishment by other means, thus presenting a grand total of 94.84 per cent. to the credit of the automatic sprinklers for successful operation in holding in check or totally extinguishing fires; the remaining 5.16 per cent. of the whole representing unsatisfactory operation of the device, the failure of which was caused either by a lack of proper water supply, serious defects in the equipment, or on account of the equipment being rendered inoperative through the cutting off of its water supply, accidentally, carelessly, or otherwise.

It may be of further interest to state from these records of the National Fire Protection Association, that 47.66 per cent. of these equipments secured their primary supply from public water service;

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34.71 per cent. were supplied from gravity tanks; 14.38 per cent. operated under pressure tank supply; 6.19 per cent. were supplied by automatic steam fire pumps; automatically operated electric pumps and connections from public fire department steamers each served as primary supply to the extent of .01 per cent.

Remarkable as is this record of success in controlling fire through the action of automatic sprinklers, it must not be assumed that there is no limitation to their power of accomplishment in fire extinguishment, as a reference to the figures above given will show that something over 30 per cent. of the fires were simply held in check by the sprinklers until other means of fighting the fire could be utilized, and this fact emphasizes the necessity for always being prepared for any untoward condition of the automatic equipment, by providing auxiliary means to control the flame when from any cause the sprinklers fail to completely extinguish it, for even with a perfectly satisfactory installation, we are often confronted with conditions due to accident or carelessness, which, for a time at least, may entirely or partially disable the equipment after it has gone into operation.

In view of the fact that you have at your command copies of the National Board of Fire Underwriters' rules and requirements covering the installation and equipment of an automatic sprinkler system, I will not burden you by a repetition of those rules, but will content myself by assuring you that when such equipment has been installed in conformity with the rules and requirements, and is kept in serviceable condition by proper supervision and care, the chance for its failure to promptly extinguish any fire in its incipiency is extremely rare, almost improbable; indeed, with proper water supply under necessary pressure behind the automatic sprinkler in almost any conceivable locality where the delivery of its spray is unobstructed, it will vindicate its past record as the most efficient fire fighting device ever conceived by the human mind; it is always in the right place at the right time, ready for any emergency and promptly performs its allotted function without dependence upon human aid or direction in its accomplishment.

I cannot dismiss consideration of the value of the automatic sprinkler as being compassed by its utilitarian qualifications as a fire extinguisher, but must call your attention to the fact that it is

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also a very potent element in the matter of saving life at the time of fire, as the promptness with which it acts in the delivery of water under the influence of heat, likewise serves as an alarm to the occupants of the threatened structures, thus admonishing them to seek safety in due time, while they are at the same moment under a drenching shower from the sprinkler which will prevent ignition of their garments, or extinguish the flame if already in evidence.

I am thoroughly convinced, as are all who are familiar with the operation of the automatic sprinkler, that if the premises of the Triangle Waist Factory, which recently burned in this city, had been under such protection, there could not have been such terrible loss of life, even as the result of wild panic, and that it is more than probable that the fire would have been completely extinguished at its incipiency and without serious loss of property.

Before giving attention to the class or specific character of fire extinguishing devices which are now generally acceptable for that purpose, it is perhaps well to call attention to the fact that as a means of fire extinguishment no agent has been discovered which can replace water under pressure and commonplace as is this knowledge, few of those who plan to use this agent seem to realize the fact that the force or pressure which serves to carry a stream to burning material is only of value when, in addition to this carrying power it includes sufficient volume to completely drown out combustion, by cutting of the supply of oxygen and cooling down the temperature of the burning material to a point below that at which it will normally ignite; force or power alone is of little value in a fire stream, hence, in planning a "layout" for fire protection, the first consideration should be given to securing a reliable and abundant water supply, and then to so design the equipment as to provide for the most effective pressure to insure the delivery of the required volume when and where needed.

In giving proper consideration to this matter of volume to be delivered, it is necessary to recall to your minds the fact that the loss of head due to friction in service pipes varies as the two-and-a-half power of the velocity; that doubling the diameter of the main quadruples its delivery capacity, and that a main which is supplied two ways or from both ends, has double the delivery capacity of one of like size when fed one way only; hence, it be-

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comes evident that in order to secure the most reliable service, the water mains in any scheme and whatever the source of supply, should be laid in complete circuit and be of sufficient size to provide volume under such head or pressure as will insure the delivery of full fire streams at each outlet in service, with a loss of head not in excess of 10 to 12 pounds below the normal pressure.

A "standard fire stream" demands the delivery of not less than 250 gallons of water per minute through a 1⅛" smooth bore nozzle and to secure this volume it requires a pressure of not less than 45 pounds to the square inch at the base of the nozzle, which will give approximately a reach of 63 feet horizontally and about 70 feet vertically.*

In considering the relative value of water supplies for fire extinguishing purposes, it is evident that a well designed public gravity system, having ample supply and pressure, should lead the list, but in the matter of supply for manufacturing plants remote from public service, the problem becomes one for special consideration in each case, with its solution dependent upon local conditions as to the source of supply and the means by which same may be best utilized, the essential point in any case being assurance of sufficient volume and reliability as to the continuance of supply under emergency demands, the quantity available at isolated plants should always be sufficient to fully serve all appliances for defence for a period of not less than one hour of continuous operation.

Private reservoirs of sufficient capacity and at an elevation to insure adequate pressure for fire service throughout the system would prove the most acceptable, but are difficult to secure and of infrequent occurrence.

An underground tank or cistern, or similar container above ground, when of suitable capacity and with means of replenishment, is a fairly satisfactory source of supply for use with steam or other power-driven pumps, when the lift is not in excess of twelve feet.

A rotary water power pump may take both its energy and supply from the head at the dam, if the supply is sufficient, but this class of pump is not always reliable for fire protection.

* The horizontal and vertical distances given are from experiments by Mr. John R. Freeman, *Transactions Am. Soc. C. E.*, XXI.

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Elevated tanks of the ordinary class and at customary elevations and capacity, are of small value for fire streams from either the standpipes or from yard hydrants.

The question of economy as a result of the uninterrupted operation of a plant being one of decided importance in plant design, it is well to bear in mind that the enforced stoppage of work caused by the occurrence of fire would be materially mitigated, if in advance of that misfortune proper precaution were taken to provide adequate and reliable means of fire protection to supplement the other essentials of good housekeeping and safe construction, and while it is undoubtedly true that the most reliable class of protection is to be found in the presence of automatic sprinklers, there are other devices and appliances which will serve good purpose in fire extinguishment when used with intelligence and judgment in that emergency, but in order to secure from such devices the most satisfactory measure of efficiency in the time of need, it is necessary that both owners and employees in every manufacturing plant should become familiarized with the appliances provided for fire fighting, and that a selected few of the most apt and intelligent of the force be formed into a fire brigade which should be drilled in the use of the appliances with such frequency and method as to render them a reliable force for the protection of the plant when fire occurs, and this result may best be accomplished by assigning to each member his appropriate position and duty in service when called upon for action, as will be found more fully detailed in the National Board pamphlet covering the formation of private fire brigades, copies of which are now in your possession.

Let me here utter a caution as to over-confidence in the matter of the efficiency of any private fire prevention equipment, however complete may be the appliances, or however well disciplined the fire brigade; in every case, upon the discovery of a fire, first call the public fire department, and then make the most prompt and intelligent use of the private appliances at command, thus insuring the concentration of all available means to conquer the flame, before it reaches proportions which may render it uncontrollable by any effort.

In giving consideration to appliances and devices now in common

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use for fire protection and not automatic in operation, time will not permit the detailed description which it is properly entitled to, and in this respect you are again referred to the pamphlets issued by the National Board, in which rules and requirements are set forth as to the construction and proper user of these manually operated appliances, which may be briefly considered as follows:

Casks and Pails.—When properly located and supplied with water, these devices are of great value in the early stages of a fire, and if intelligently used, frequently prevent serious loss.

Chemical Extinguishers.—Like fire pails, these devices are of decided value when available, and have the advantage of enabling the operator to direct an efficient stream immediately upon the seat of combustion up to a distance of about 40 feet.

Vertical Pipes (*Standpipes*).—When located in accessible positions and under proper head or pressure, with hose and nozzle attached, serve well in the control of more advanced fires, provided the operator retains control of his facilities sufficiently to "stay by" the appliance and intelligently direct the stream delivered.

No stand pipe less than three inches in diameter should be installed; having this diameter, it will supply two streams through two inch hose with nozzles of ⅜ or ¾ inch diameter, and four streams with same sized nozzles, through 1½ inch hose. Standpipes supplied from elevated tanks of usual capacity and at average elevations, are not of value for hose stream service.

Steam Jets.—Are sometimes quite efficient in suppressing fires in dry-rooms and other confined spaces; where used there should be ample boiler capacity behind them to insure volume of steam.

Sand Pails.—Dry, granular sand, freed from excess of clay or loam, forms a very efficient means of extinguishing fires in oils, varnish and other inflammable fluids by smothering the flame. A proper supply should be maintained, with a scoop for its distribution, in places where such inflammables are used or stored.

Steam Fire Pumps.—Are a very efficient means of fire protection when they are properly designed for the special purpose, and situated in a fireproof structure separated from the general plant, with independent and duplicate sources of steam supply, and have a water supply of sufficient capacity to insure continuous operation for a period of not less than one hour's duration at full capacity.

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Ordinary trade pumps, while efficient for general service, are seldom reliable under the stress of fire demands.

Centrifugal or Turbine Pumps.—When these are designed especially for duty in fire service they promise to be of considerable value when properly located as to exposure and fitted with both steam and water supply required for the direct-acting steam fire pump.

Electrical Pumps.—Special fire pumps operated by the electric current are coming into use to a considerable extent, and when properly designed and located, with two separated and reliable sources of supply for power, may be expected to do good service.

Rotary Pumps.—Water-power rotary fire pumps when designed for the purpose are capable of rendering very good service under favorable conditions of power supply and location, but, as a rule, the best results are not obtainable, for the reason that the usual location of the device is in a wheel pit or other equally inaccessible spot difficult to reach in case of fire, and for the same reason is liable to be neglected as to inspection and up-keep.

Fire Hydrants.—Whether public or private, should conform in construction to the National Standard, with a barrel of not less than six (6″) inches in diameter, and be fed from a service main of not less than of the same dimensions forming a complete circuit of the system, without dead ends, meter connections or other obstructions to the free flow of water, and be located at a distance of not less than 50 feet from the buildings to be protected.

Hose connections or outlets should be of the National Standard pattern, unless the city outlets differ from it, in which event the outlets on private equipment should conform to the latter.

Fire Hose.—This is an important item controlling the efficiency of fire streams where hose is used, from the fact that the loss of head due to friction in the passage of water through hose of the very best rubber lined quality (of 2½″ size) amounts to about 14 pounds per hundred feet of hose, while with the ordinary quality of hose on the market and in use, such loss of head may reach 25 or more pounds per 100 feet.

Hose for use on hydrants and other devices supplying streams on the outside of buildings should be of not less than 2½″ diameter, with nozzles of 1⅛ inch smooth bore pattern, while hose for use

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inside of a structure should preferably be not in excess of 2 inches in diameter with a 7/8 inch smooth nozzle, in order to insure quick and effective handling by novices.

Underground Piping.—Should be laid in complete circuit, and where the system is extensive, should be gridironed in order to secure circulation. Pipe sizes of less than six inches in diameter should not be permitted in any equipment. One steam fire engine connected to a six inch service line will cut off supply to any other device in the same line of service.

In order to be assured of even an approximately satisfactory system for any individual plant, practically none of the appliances and devices above mentioned should be omitted, except in the matter of fire pump selection, where a choice may be had, and the presence of all such apparatus is deemed essential to supplement the protection to be expected from a properly installed automatic sprinkler system.

If the time allotted to me for delivery of this address would permit the effort, I might go somewhat into detail as to the principal causes of fires in manufacturing plants, but this subject is one with a very broad field of possibilities and of experience, hence, I must content myself by stating that as about 80 per cent. of all fires are due to carelessness, and that in the final analysis of the remaining 20 per cent. which is usually attributed to accident, close investigation would show that as about 80 per cent. of these so-called accidents are justly attributable to a lack of proper supervision of machines or devices and of process methods, we will have left but a small percentage which may be classed as entirely accidental and unpreventable, and a faithful adherence to the gospel of "good house-keeping" would unquestionably prove the cure for even the supposedly unpreventable causes.

In concluding my remarks I desire to call your attention to the value of the text books, or "Rules and Requirements" published by the National Board of Fire Underwriters, covering the fire hazards and means of fire prevention or extinction, copies of each of which it has afforded me much pleasure to present to you, and to express the hope that they may prove of service to you in your work of standardizing factory design in the matter of fire prevention and fire protection, and thus, by instruction and guidance, lead the unit

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of values into such comprehension of its importance to the conservation of the interests of the whole people of this country, as to show marked results in the reduction of the inexcusable fire waste, which, if unchecked, promises to certainly impoverish our people as a whole, and to you, young gentlemen, comes the call to "spread the gospel of conservation of our created resources," and I trust your response may be both earnest and successful.

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ROASTING OF NICKEL-COPPER MATTE.*

By EDWARD F. KERN and H. W. WALTER.†

While there are numerous methods for the recovery of nickel from ores and mattes, roasting at some stage in the process is common to all, hence a better knowledge of the roasting of these materials is of much practical importance.

The development of the metallurgy of nickel has been of a secretive nature and practically nothing has been published of recent years which discloses the trend of experimentation. After looking through the literature on the subject, the things which were suggested for investigation are the following:

(*a*) What temperature and conditions of roasting of a nickel matte will give the greatest amount of oxidation?

(*b*) What temperature and conditions of roasting of a nickel matte will cause the production of the greatest amount of nickel sulphate?

(*c*) What is the temperature of decomposition of nickel sulphate?

(*d*) What temperature and conditions of roasting of a nickel-copper matte will give the maximum oxidation of the nickel sulphide and the minimum oxidation of the copper sulphide, or vice versa?

(*e*) What is the temperature at which reaction takes place, and what are the products which result when the following mixtures are heated: nickel oxide and cuprous sulphide; cuprous oxide and nickel sulphide; nickel sulphate and copper oxide; copper sulphate and nickel oxide?

Abstracts of Literature.

In the roasting of copper sulphide materials, considerable careful work has been done in connection with the Zeirvogel Process, which depends upon controlling the temperature and conditions of

* Article based on an investigation conducted in the metallurgical laboratory of the School of Mines, Columbia University, New York City.

† Assistant professor, and Met.E. 1909, respectively.

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roasting so that the sulphides of iron and copper are oxidized to their respective sulphates. Then by raising the temperature of the furnace sufficiently to decompose these sulphates into their oxides and sulphur trioxide, the latter reacts with the sulphide of silver and produces silver sulphate, which may subsequently be recovered by leaching.* The operation of roasting, preliminary to leaching out the silver sulphate, is obviously a more or less delicate one, and before the advent of the pyrometer required skilled labor, but with the aid of a pyrometer and a knowledge of the temperature of decomposition of the sulphates of iron, copper and silver, the process could at the present time be carried on to better advantage. The process, at present, is of no commercial importance, but is cited as it has some bearing on the subject of controlling the formation of sulphates during roasting of sulphide materials.

R. H. Bradford, a graduate student in Columbia University in 1901, determined the temperature of decomposition of the following:† $FeSO_4$ at 590° C.; $CuSO_4$ at 650° C. to 655° C.; $CuO.CuSO_4$ at 702° C.; Ag_2SO_4 commences at 860° C. and is rapid at 1,095 ° C. These determinations were made by heating the respective dehydrated sulphates in a small, accurately controlled electric furnace. By noting the temperature at which there was a retardation in the progressive rise of temperature (which indicated an absorption of heat as the result of that required to decompose the compounds) the temperature of decomposition was found.

Felix Warlimont conducted a research on the treatment of a copper-nickel sulphide ore, and found it possible to so control the temperature of roasting that the iron sulphides were oxidized to Fe_2O_3, the copper sulphide to $CuSO_4$, and the sulphide of nickel remained unaltered.‡ This affords a method whereby copper and nickel can be separated and subsequently recovered from sulphide ores. He used for his experiments an ore which contained 25.25 per cent. S, 42.40 per cent. Fe, 0.75 per cent. Cu, 3.10 per cent. Ni, and 13.11 per cent SiO_2. A number of roasts were conducted under varying conditions, as regards temperature and time, which showed that by holding the temperature between 450° C. and

* *Mineral Resources*, 1876, p. 379. *Trans. Am. Inst. Min. Engs.*, 1876, p. 285; 1903, p. 50. Schnabel's "Handbook of Metallurgy," ed. 1905, I., p. 899.

† *Trans. Am. Inst. Min. Engs.*, 1903, p. 50.

‡ *Metallurgie*, 1909, 6, Heft 3, pp. 83-92; Heft 4, pp. 127-134.

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480° C. for nine hours, then raising it to 550° C. and maintaining it there for four hours longer, the copper sulphide was converted into $CuSO_4$, the iron sulphides into Fe_2O_3 and the greater part of the nickel sulphide was not altered. The author proposed a method whereby copper-nickel sulphide ores could be treated and the copper recovered by leaching the roasted ore, and the nickel obtained from the leached ore by smelting to a nickel matte, which then treated electrolytically would yield pure nickel.

In Schnabel's "Handbook of Metallurgy," ed. 1907, II., on pages 629 to 632 are given the following reactions:

(*a*) Nickel oxide does not react with its sulphide at high temperatures.

(*b*) Nickel oxide heated with iron sulphide gives nickel sulphide and iron oxide; in the presence of silica, iron silicate forms.

(*c*) Nickel oxide does not react with copper sulphide when heated.

(*d*) Nickel monosulphide when heated in the electric furnace breaks down to the subsulphide (Ni_2S). The subsulphide does not exist in mattes.

(*e*) Nickel sulphide at high temperatures reacts with copper oxides, producing nickel oxide and copper sulphide.

(*f*) Nickel sulphide does not react with iron oxides when heated.

(*g*) Nickel sulphide at red heat is slowly and completely oxidized by steam.

(*h*) By carefully roasting compounds, or mixtures, of nickel, copper and iron sulphides, the iron will be oxidized to oxides, and the greater part of the copper and nickel converted into sulphates. As nickel sulphate is a more stable compound than copper sulphate, it is possible with sufficient sulphur in the ore, to conduct the roasting so that the nickel remains chiefly as sulphate, and the copper as oxide.

(*i*) Metallic nickel, in some cases, is more easily oxidized than nickel sulphide.

(*j*) Water vapor has but little effect on copper sulphide at red heat, whereas at white heat, metallic copper and hydrogen sulphide results.*

* Schnabel's "Handbook of Metallurgy," ed. 1905, I., p. 12.

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Formerly it was thought that when mattes, containing iron, copper and nickel, were subjected to oxidizing conditions, such as roasting or converting to blister-copper, the iron sulphide was first oxidized to the exclusion of the sulphides of copper and nickel, after which the nickel sulphide was oxidized leaving the copper sulphide unaltered. That is, the nickel in copper-nickel mattes was regarded as an element which replaced iron, and during the converting of such a matte the nickel was oxidized and slagged similar to iron, leaving the copper sulphide as such. This conception has recently undergone a complete change, as the result of the publication of an article by D. H. Brown on "The Behavior of Copper-nickel Mattes in the Bessemer Converter."* The conclusions drawn after a long series of experiments on the converting of copper-nickel matte are as follows:

1. Nickel is not an element replacing iron in mattes.

2. Nickel-copper alloys act in the "matte-blow" like one metal.

3. Nickel-copper alloys follow, during the "matte-blow," exactly the same laws as govern the behavior of copper alone.

4. Nickel does not, as previously supposed, follow iron into the slag, but displays a reluctance to part with the copper, so much so that one pound of nickel passing into the slag drags one and a quarter pound of copper with it.

5. While nickel during the last of the "blow" does oxidize, yet it lags behind sulphur as a heat producer. It is impossible to "blow" all the nickel out, about 2 per cent. is always left in the finished copper.

6. The final product of blowing a copper-nickel matte is one homogeneous metal, no matter whether high copper and low nickel, or vice versa.

7. The nickel exists in the matte as the sub-sulphide Ni_2S, and not as the monosulphide NiS.

Epitome of Literature.

Although the converting of matte is an operation differing greatly from roasting, it is however accomplished as the result of selective oxidation. The reactions taking place during converting are at temperatures above the melting point of the matte, whereas

* *Trans. Am. Inst. Min. Engs. Bulletin*, No. 40, April, 1910, pp. 285-305.

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during roasting, the reactions occur at temperatures between 400° C. and 850° C. Still, in both cases, the order in which the constituents of the matte are selectively oxidized are generally the same; so if there are any laws which may be applied to roasting, the same may also be applied to converting, and vice versa.

EXPERIMENTAL: PART I.

Decomposition Temperature of the Sulphates of Iron, Copper and Nickel.

Before starting the roasting experiments, the decomposition temperatures of $FeSO_4$, $CuSO_4$, and $NiSO_4$ were determined by applying the principle of the air thermometer; which is, that with each degree rise in temperature, the expansion in volume of a gas is proportional to its absolute temperature (Law of Charles).

The apparatus used for these determinations was a small porcelain tube electric furnace, heated by passing a current through platinum wire wrapped around the middle. Both ends of the tube were sealed with rubber stoppers. Through one of the stoppers was sealed the terminals of a LeChatelier thermo-electric couple, which was sufficiently long so that the junction rested on the center of the tube. Sealed into the other stopper was a quarter-inch glass tube which was connected, by means of heavy-walled, pressure, rubber tubing to a manometer filled with ethyl alcohol. The apparatus is shown in the photograph. The size of the porcelain tube was 18 inches long, and inside diameter three quarters of an inch. It was wrapped about the center with 14 feet of No. 23 gauge platinum wire (260 grams), connected to binding posts with which connection with the 115-volt lighting circuit was made. The current was controlled by means of a rheostat. The manometer was made by bending a five-feet length of glass tubing, 5 mm. bore, forming a U tube with one leg 33 inches long, the other leg 20 inches long and a right angle bend of 6 inches, to which connection with the porcelain tube was made. Between the manometer and the porcelain tube was inserted a glass T tube, to one leg of which was sealed a stop-cock, the purpose of which was to release the air from the system during the time the porcelain tube was being heated to the desired temperature (70° C. to 80° C. below the decomposition

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temperature). By the side of the 33-inch leg was placed a 350-millimeter scale.

The operation of determining the decomposition temperature of the sulphates was as follows: About 5 grams of the dehydrated salt was placed in a porcelain boat, which was then shoved to the center of the porcelain tube. The thermo junction was imbedded in the salt, so as to be in contact with the bottom of the boat, as that was the part which was hottest during the heating. The cur-

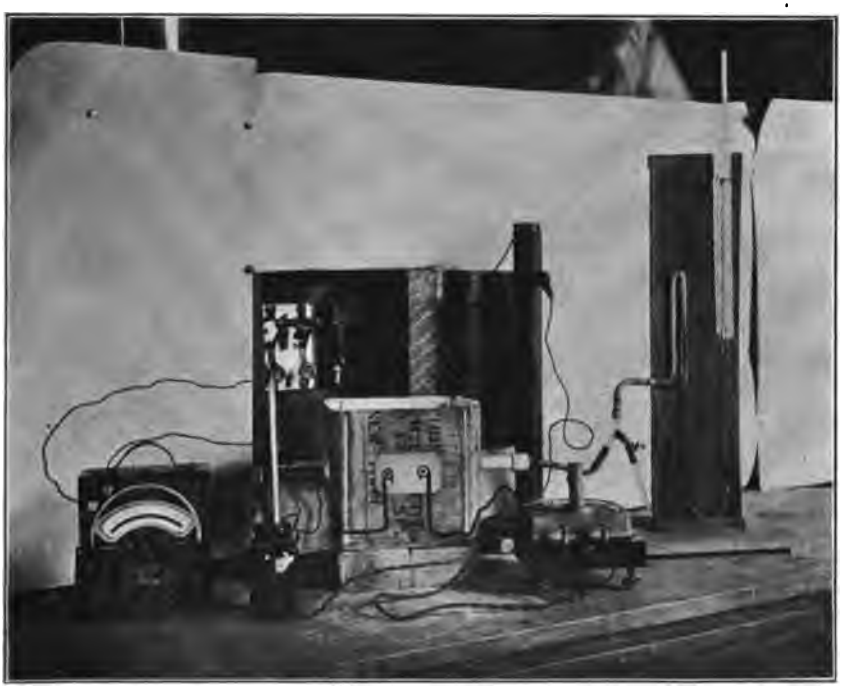

FIG. 1.

rent was then turned on, and as the tube heated up the inclosed air expanded and caused the column of alcohol to rise regularly in the graduated leg of the manometer—when the release cock was closed. The deflection of the galvanometer and the position of the alcohol column were simultaneously read, and so long as there was no decomposition of the salt, the rise of the manometer column proceeded quite uniformly with the rise in temperature; but when the temperature of composition was reached, there was an evolu-

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tion of So_3, which addition of gas to the system caused a sudden and rapid rise of the alcohol column. A number of determinations were made on the dehydrated salts of $FeSO_4$, $CuSO_4$ and $NiSO_4$. In each case, a preliminary run was made in order to find the approximate temperature at which the decomposition occurred, so that the release cock could be allowed to remain open until the temperature about 75° C. below that of decomposition was reached. This was done because of the delicacy of the manometer, as a rise in temperature of 10° C. gave a rise in the manometer column of from 8 to 15 millimeters. On reaching the desired temperature, the release cock was closed and the readings were recorded. The data obtained is given in Tables I., II. and III., and is shown by curves no. I., no. II. and no. III.

TABLE I.

DECOMPOSITION TEMPERATURE OF $FeSO_4$.

Galvanometer Deflection, M.V.	Corrected Temperature, °C.	Manometer Column, mm.	Rise of Column, Interval.	Galvanometer Deflection, M.V.	Corrected Temperature, °C.	Manometer Column, mm.	Rise of Column, Interval.
48	517	—	(—)	48	517	52	—
49	526	23	—	49	526	68	16
50	535	26	3	50	535	80	11
51	544	33	7	51	544	78	—2
52	553	40	7	52	553	75	—5
53	562	46	6	53	562	73	—2
54	571	51.5	6.5	54	571	76	+3
55	580	57.5	6	55	580	80	4
56	589	65	7.5	56	589	86	6
(57)	(598)	75	(10)	(57)	(598)	92	(6)
58	607	88	13	58	607	102	10
59	616	109	21	59	616	113	11
60	625	123	—	60	625	126	13

In Table I. are given the records obtained when dehydrated ferrous sulphate was heated. The dehydration of the $FeSO_4 \cdot 7H_2O$ was done by heating the salt on a water-bath for about two hours, then reheating in an assay muffle furnace for about two hours at a temperature between 225° C. and 250° C. During the period that the dehydrated salt was being heated up in the tube for the determinations of the decomposition temperature—a depression of the manometer column took place before decomposition occurred. This depression is accounted for by the fact that on opening the tube, after the run, a small amount of moisture was seen on the colder part of the tube, which explained the depression of the column between 450° C. and 500° C. The moisture was the result

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TABLE II.

DECOMPOSITION TEMPERATURE OF $CuSO_4$.

Galvanometer Deflection, M V.	Corrected Temperature, °C.	Determination No. 1.		Determination No. 2.		Determination No. 3.		Determination No. 4.	
		Manometer Column, mm.	Rise of Column, Interval.	Manometer Column, mm.	Rise of Column, Interval.	Manometer Column, mm.	Rise of Column, Interval.	Manometer Column, mm.	Rise of Column, Interval.
53	566	29	—	34	—	28	—	24	—
54	575	34	5	42	8	34	6	28	4
55	584	39	5	49	7	39	5	32	4
56	593	43.5	4.5	57	8	45	6	36	4
57	602	48.5	5	65	8	51	6	40	4
58	611	53.5	5	74	9	56.5	5.5	44.5	4.5
59	620	59	5.5	82	8	62.5	6	49	4.5
60	629	64	5	90	8	68.5	6	53.5	4.5
61	638	69	5	99.5	9.5	75	6.5	58.5	5
62	647	75	6	109	9.5	82	7	64.5	6
(63)	(656)	84	(9)	123	(14)	94	(12)	73	(8.5)
64	665	93	9	137	14	106	12	82	11
65	674	104	11	153	16	118	12	93	11
66	683	114	10	170	17	132	14	105	12
67	692	119	15	—	—	—	—	116	11

TABLE III.

DECOMPOSITION TEMPERATURE OF $NiSO_4$.

Galvanometer Deflection, M.V.	Corrected Temperature, °C.	Determination No. 1.		Determination No. 2.		Determination No. 3.		Determination No. 4.	
		Manometer Column, mm.	Rise of Column, Interval.	Manometer Column, mm.	Rise of Column, Interval.	Manometer Column, mm.	Rise of Column, Interval.	Manometer Column, mm.	Rise of Column, Interval.
63	655	20	—	36	—	47	—	32	—
64	664	23	3	41	5	53	6	36	4
65	673	26	3	46	5	58	5	41	5
66	682	29.5	3.5	51	5	63	5	45	4
67	691	33.5	4	56	5	68.5	5.5	50	5
68	700	38	4.5	62	6	75	6.5	55	5
69	709	43.5	5.5	68	6	80.5	5.5	60	5
70	718	49.5	6	72	4	86.5	6	65	5
71	727	55.5	6	78	6	92.5	6.5	71	6
(72)	(736)	67	(11.5)	86	(8)	(101)	(8.5)	(81)	(10)
73	745	79	12	94	8	109	8	92	11
74	754	94	15	106	12	120	11	105	13
75	763	115	21	123	17	132	12	129	14
76	772	—	—	144	21	150	18	145	16

of incomplete dehydration of the ferrous sulphate, even at 225° C. to 250° C. The condensing of the trace of water vapor on the colder part of the tube caused a decrease in volume of the air and vapor, bringing about a depression of the manometer column. A second depression also occurred between 510° C. and 570° C., but not at identical ranges of temperature any two times. This second

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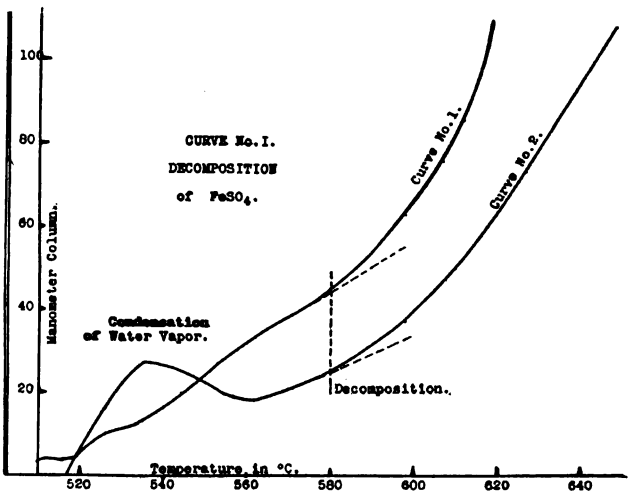
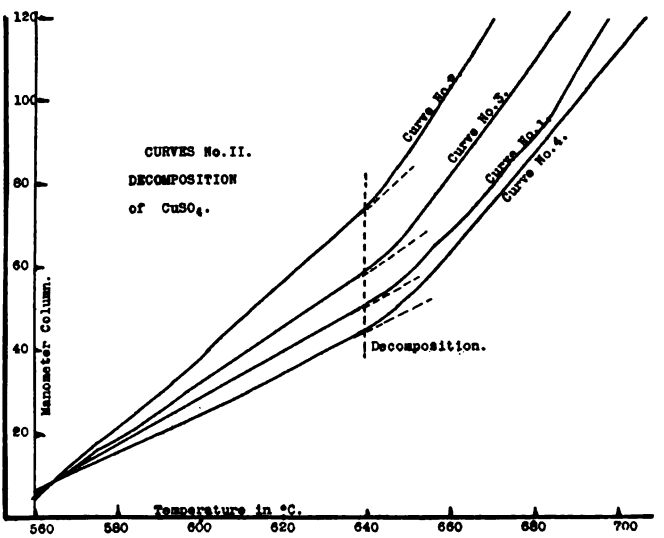
7a.



CURVE I.

8a.



CURVE II.

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depression is explained by the fact that the dehydrated salt was partially oxidized, forming a basic sulphate as is shown by the equation: $3FeSO_4 + O = Fe_2(SO_4)_3 \cdot FeO$. The oxygen which took part in the oxidation caused a decrease in volume of the air in the tube. The depression did not occur within the same temperature range in the duplicate runs, because of the difference in the rate of time of heating up the tube. In order to find out at what temperature the basic sulphate began to form, a small amount of the dehydrated salt was heated up slowly, and the appearance noted from time to time. It was found that the salt began to assume a reddish-brown color when the temperature had reached 280° C., and that it oxidized more rapidly with rise of temperature, and was quite rapid above 500° C. Even though the depressions of the manometer column did not duplicate each other, the decomposition temperature was in each case the same, starting at about 580° C. and taking place rapidly at 598° C., and above.

In Table II. is recorded the determinations of the decomposition temperature of copper sulphate. The $CuSO_4 \cdot 5H_2O$ was dehydrated by heating the salt on a clay roasting dish in an assay muffle furnace for about two hours at a temperature between 325° C. and 375° C. A slight depression of the manometer column occurred during the determination of the decomposition points, when the temperature of the tube reached about 475° C., but in no case was it so marked as with the iron sulphate. As basic sulphates of copper are not produced by the direct oxidation of the normal sulphate, but are formed by decomposition at high temperatures, other causes were sought to explain the slight depression of the manometer column at about 475° C. It was then found to be due to the condensation of a small amount of water vapor on the colder parts of the tube. The decomposition of the copper sulphate began slightly when 647° C. was reached, and was markedly rapid at 656° C., and above.

In Table III. are given the data obtained for the decomposition temperature of dehydrated nickel sulphate. The dehydration of the $NiSO_4 \cdot 6H_2O$ was done in the same manner and at the same temperature as that of the $CuSO_4 \cdot 5H_2O$. A slight depression of the manometer column also occurred, during the heating of the dehydrated salt in the tube, and within the same temperature range

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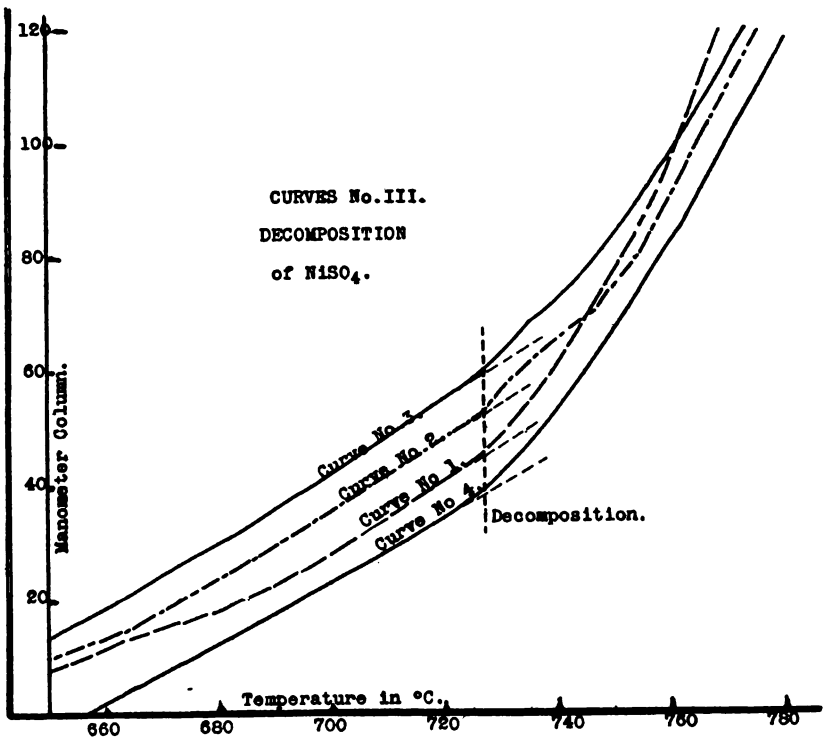
as that of the dehydrated copper sulphate. As no basic sulphates of nickel are formed as the result of direct oxidation of $NiSO_4$, the depression was not due to the absorption of oxygen, but to the condensation of a small amount of water vapor on the cold parts

8b.



CURVE III.

of the tube. The decomposition of nickel sulphate was found to start slowly at 727° C., and to take place rapidly at and above 736° C.

EXPERIMENTAL: PART II.

Determination of the Freezing Curves of the Sulphides of Nickel and Copper, and of Mixtures of the Two.

The nickel sulphide was prepared by mixing 306 grams of pure nickel oxide with about 600 grams of sulphur, placing the mixture

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in a Dixon graphite crucible, and covering it with charcoal. The crucible was then heated at a temperature slightly above the boiling point of sulphur (B.P. of S = 445° C.) until sulphur ceased to be given off. The temperature of the furnace was then raised to about 900° C., and the mass maintained in molten condition for a few minutes, after which it was poured into a dry clay crucible, and allowed to cool. A sample was taken and analyzed, showing it to contain 69.5 per cent. Ni, which corresponds to the formula Ni_5S_4, which theoretically contains 69.6 per cent. Ni.

The copper sulphide was prepared in a manner similar to that of the nickel sulphide, by heating pure copper oxide with an excess of sulphur, and afterwards bringing the mixture to fusion. It contained approximately 80 per cent. Cu, corresponding to Cu_2S.

The determinations of the freezing-points were made by placing about 15 grams of the materials in small porcelain crucibles, covering it with a small amount of sulphur and charcoal, and heating to fusion in a gas-fired "Dental" furnace (size inside: 3 in. deep by 2.5 in. diam.). The temperatures of the molten mass during cooling were taken by means of a LeChatelier thermo-electric pyrometer, by placing the platinum couple (protected by a clay tube) into the molten sulphide, and then turning off the gas and blast. During the cooling period, the galvanometer deflections and time were simultaneously read, and the results plotted on coördinate paper, making galvanometer deflections the ordinates, and time in minutes the abscissa, which are the curves: IV., *a, b;* V., *a, b;* VI., *a, b;* VII., and VIII.

Five freezing-curve determinations of the following were made: (1) Cu_2S; (2) Ni_5S_4; (3) mixture of 64 parts Cu_2S + 36 parts Ni_5S_4; (4) mixture of 50 parts Cu_2S + 50 parts Ni_5S_4; (5) Mixture of 40 parts of Cu_2S + 60 parts Ni_5S_4.

The freezing points were found to be the following:

1. Copper sulphide (Cu_2S) freezes at 1110° C.

2. Nickel sulphide (Ni_5S_4) freezes at 790° C.

3. Mixture of 64 per cent. Cu_2S + 36 per cent. Ni_5S_4 starts to freeze at 948° C. when the excess of Cu_2S begins to solidify, then at 740° C. the eutectic freezes.

4. Mixture of 50 per cent. Cu_2S + 50 per cent. Ni_5S_4 starts to freeze at 899° C. when the excess of Cu_2S begins to solidify, and the eutectic freezes at 738° C.

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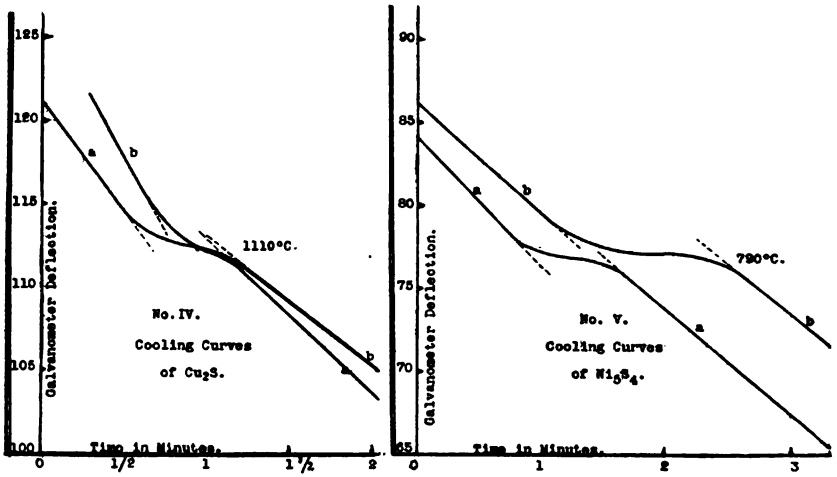
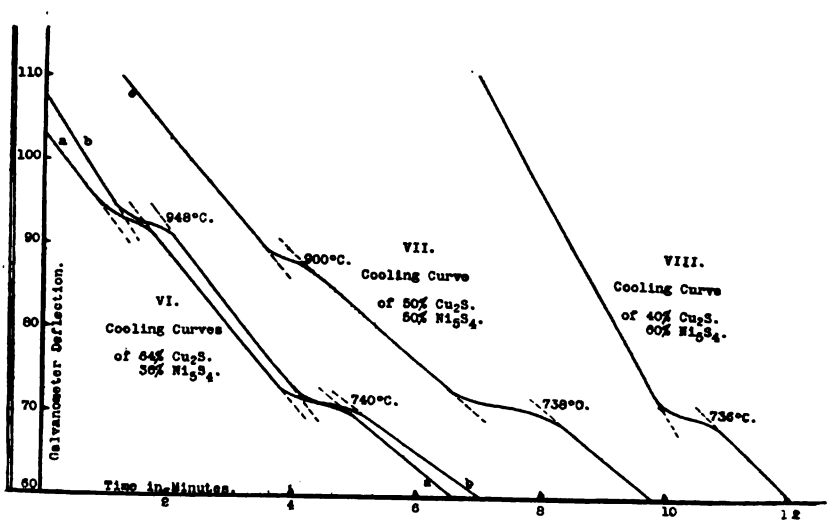
11a.



CURVES IV AND V.

11b.



CURVES VI, VII AND VIII.

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5. Mixture of 40 per cent. Cu_2S + 60 per cent. Ni_5S_4 freezes at 736° C. During the cooling of this mixture, no retardation of the fall in temperature was noted, except at this one point, which indicates that this mixture is in the eutectic proportion. With each of the other mixtures, a retardation was also noted at approximately this same temperature, so it is without doubt that the eutectic of $Cu_2S + Ni_5S_4$ is in the proportion of this mixture.

These freezing curves were determined because during the roasting of mattes the particles of material are apt to fuse together, if the furnace is allowed to become too hot at the start of the roast, before the particles are coated with a layer of oxide. By knowing the temperature at which the constituents of the matte begin to fuse, the condition of the roasting may be better regulated in order to prevent sintering, thus obtaining a more rapid and a more complete oxidation.

Experimental: Part III.

Method for the Analysis of Roasted Nickel-copper Matte.

In order to determine the condition of the roasting of the matte and the amount of oxidation of the nickel and copper constituents at regular intervals of the roast, it was necessary to have a method of analysis which would give a separation of the sulphates, the oxides, and the sulphides of nickel and copper from each other, but no such method was found in the litetrature.

There is no difficulty in the separation of nickel from copper in solutions, but to get a separation of the oxides of nickel and copper when associated with their sulphides was found to be a very difficult problem. Having a roasted matte containing $NiSO_4$, $CuSO_4$, NiO, CuO, Ni_2S, and Cu_2S, the separation of the sulphates from the oxides and sulphides is readily accomplished by leaching the sample with boiling water, leaving a residue of the oxides and sulphides of nickel and copper. The usual method of analysis of roasted copper matte for the separation of CuO from Cu_2S is to leach the material for about 20 minutes with a 10 per cent. HCl solution (by vol.) at slow boiling, which dissolves the CuO but does not attack the Cu_2S. This solvent, however, cannot be used for separating the oxides of nickel and copper from their sulphides, as sulphide of nickel is soluble in this acid liberating H_2S,

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which precipitates copper from the solution. It was thought that possibly a more dilute acid would not dissolve the sulphide of nickel, but still take the oxides of both nickel and copper into solution. With this in view, samples of raw nickel-copper matte were treated with 1 per cent. and 5 per cent. solutions (by volume) of hydrochloric acid, and also with dilute sulphuric acid of these same strengths, but in all cases the nickel sulphide constituent of the matte dissolved, and H_2S was liberated. It was then suggested that an ammoniacal solution be tried, as copper oxide is soluble in ammonia, and in ammonia solutions containing ammonium salts.* Samples of the raw matte were then treated for one hour with concentrated ammonia (sp. gr. 0.9), and also with concentrated ammonia saturated with ammonium chloride crystals, in both cases at boiling. The solutions were tested for nickel and copper, which showed that these solutions do not attack raw matte. Next the ignited oxides of nickel and of copper were treated in the same manner, with the result that it was found that both Cu_2O and CuO are soluble, and NiO is not soluble in these solutions.

In order to determine whether all the copper oxide in roasted nickel-copper matte is soluble in ammonia solutions, some of the matte was roasted. It was then first treated with boiling water to remove the sulphates, and the residue digested at boiling for about an hour with concentrated ammonia saturated with ammonium chloride. After filtering and washing, the residue was redigested with 100 c.c. of hot 10 per cent. solution (by volume) of hydrochloric acid, in order to see if any copper oxide remained after the ammonia solution leach. The acid solution contained no copper, which suggested that possibly ammonia solutions containing ammonium chloride could be used in the analysis of the roasted matte. Concentrated ammonia diluted with an equal volume of water was next tried, the leaching being conducted in the same manner as described, with the result that all the copper oxide was dissolved, whereas the nickel oxide, the copper sulphide and the nickel sulphide were not attacked by a solution of this strength.

With these data, attempts were made to devise a method of analysis for nickel-copper mattes of various stages of oxidation. A number of analyses were run, first leaching the one gram

* Schnabel's "Handbook of Metallurgy," ed. 1905, I., p. 9.

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samples with 175 c.c. of boiling water to remove the sulphates, then treating the residue at slow boiling for half an hour with 150 c.c. of dilute (1:1) ammonium hydroxide to which was added 7.5 grams of NH_4Cl crystals to remove the copper oxide, and finally dissolving the residue in 15 c.c. conc. HNO_3 then evaporating to SO_3 fumes with 8 c.c. of conc. H_2SO_4, giving a solution containing the nickel which was in the samples as oxide and sulphide and the copper as sulphide. These three solutions were analyzed for nickel and copper, they being separated by adding strips of metallic aluminium and boiling which precipitates the copper and not the nickel. The precipitated copper was dissolved in nitric acid, the solution neutralized with ammonia and titrated with standard potassium cyanide solution. The nickel solution was titrated with standard potassium cyanide solution, as outlined later on. By this method of procedure it was hoped that the respective amounts of $CuSO_4$, $NiSO_4$, CuO, NiO, Cu_2S and Ni_2S could be determined, but the results were so erratic that it was found necessary to determine merely the $CuSO_4$, $NiSO_4$ and the S. The estimation of the nickel oxide by the ammonia leach method, was to determine the S in a duplicate sample, subtract the amount of S combined as sulphates, then allot the apportionate amount of the remaining S to Cu and Ni in the ratio of Cu_2S and Ni_2S in the unroasted matte, the remaining nickel then considered as NiO. As the matte was not composed wholly of Cu_4S and Ni_2S, but also contained a metallic alloy of copper and nickel (seen under the microscope) this method of analysis could not be used.

Determination of Sulphide Sulphur.

One gram samples were first leached with boiling water to remove the sulphate sulphur. The residues, after washing, were digested at temperature not over 85° C. with 15 c.c. of concentrated HNO_3 saturated with $KClO_3$, until all the sulphur was dissolved. (If the digestion is made at boiling, part of the S will separate as small globules, making it difficult to get into solution.) The solutions were then evaporated to dryness in water-bath, about 5 c.c. of conc. HCl added, and again evaporated to dryness. The dry masses were taken up with about 5 c.c. of conc. HCl and 15 c.c. of water and heated to boiling, after which the residues

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were filtered off and washed with hot water. The filtrates were brought to boiling, 2 grams of barium chloride crystals added and the precipitates allowed to settle. The solutions were filtered, the precipitates washed free of chlorides with hot water, dried, ignited in a porcelain crucible at dull red heat, a drop of conc. HNO_3 added, and the crucible again heated over flame of a Bunsen burner. The weight of the $BaSO_4$ multiplied by 0.1373 gives the S equivalent.

Separation of Copper and Nickel in Solution.

The separation of copper from nickel, when together in solution, was accomplished by boiling with sheet metallic aluminium. The conditions necessary for complete separation are: the solution must be either sulphate or chloride, containing between 2 and 4 c.c. of free H_2SO_4, or HCl, per 100 c.c. solution, and boiled till all copper precipitated (test by adding a few drops of H_2S water; no coloration indicates no copper in solution). The solution then decanted from the precipitate and the aluminium, which washed twice by decantation. The solution, containing the nickel was titrated with standard KCN solution, as outlined. The copper precipitate was brought into solution by adding about 8 c.c. conc. HNO_3 and 10 c.c. water, and heating, the aluminium strips not being removed from the beaker (metallic aluminium is not soluble in HNO_3). After the copper was dissolved, the solution was poured into another beaker, and the strips of aluminium washed twice with water. The copper solution was titrated with standard KCN solution as outlined.

Titration of Copper Solutions.

The Cyanide Titration Method, as given in most text-books on analysis,* was used: The solution containing the copper as sulphate or as nitrate is neutralized with ammonia and about 5 c.c. added in excess. The solution is cooled, diluted to about 250 c.c., and titrated to colorless end-point with a standard potassium cyanide solution (22 grams of KCN per liter). The standard is obtained by dissolving 0.2 gram of copper foil in nitric acid, boil off red

* Sutton's "Volumetric Analysis," 1904 ed., p. 190. Rickets and Miller's "Notes on Assaying," 1897 ed., p. 164. Low's "Technical Methods of Ore Analysis," 1908 ed., p. 90.

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fumes, neutralize with ammonia and 5 c.c. added in excess; then dilute to about 250 c.c. and titrate to colorless end-point.

Titration of Nickel Solutions.

The Moore Cyanide Titration Method* was used for determining the nickel. It is as follows: The solution, containing about 0.1 gram of nickel, either as sulphate, nitrate or chloride, and sufficient free acid to prevent the formation of a precipitate on subsequent addition of ammonia, is rendered alkaline with ammonia, but not over 1 c.c. in excess. If a precipitate forms, add a small amount of ammonium chloride. Dilute to 200 c.c., add 1 c.c. of potassium iodide solution (10 grams per 100 c.c.), cool to ordinary temperature, then add exactly 1.0 or 2.0 c.c. of standard $AgNO_3$ solution (3.0 grams per liter), which gives a turbid yellow solution. Titrate directly with a standard KCN solution (22 grams per liter) till end-point is the clearing of the turbidity, then add the standard $AgNO_3$ solution, drop by drop, till opalescence appears (precipitation of AgI). The respective volumes of standard $AgNO_3$ and KCN solutions are read off; the KCN value of the $AgNO_3$ solution which added, is deducted from the total value of the KCN solution used, the difference is the number of c.c. of the KCN solution required by the nickel. The metallic silver value of the KCN solution multiplied by 0.27193 gives the amount of nickel in soluiton (2Ag: 1Ni):

$$NiSO_4 + 4KCN = Ni(CN)_2 \cdot 2KCN + K_2SO_4,$$
$$AgNO_3 + 2KCN = AgCN \cdot KCN + KNO_3.$$

The standard of the solutions may also be obtained by taking a weighed amount of pure $NiSO_4 \cdot 6H_2O$, dissolving it in water and proceeding as above.

Nickel may be accurately determined by this method, when aluminium, ferric iron, or magnesium, are present, if 5 to 10 grams of either citric acid or tartaric acid are added before neutralizing the solution with ammonia, in order to prevent the precipitation of these metals. Copper, manganese and cobalt, must be absent. The copper is previously separated either by

* Sutton's "Volumetric Analysis," 1904 ed., pp. 251–254. Low's "Technical Methods of Ore Analysis," 1908 ed., p. 324. Johnson's "Chemical Analysis of Special Steels, etc.," 1909 ed., p. 104.

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H_2S, or by means of aluminium foil. Cobalt may be regarded as nickel. The most harmful element is sulphur when present as H_2S, which darkens the solution; it may be removed by agitating the solution with PbO, or Bi_2O_3.

The relation of the $AgNO_3$ solution to the KCN solution must be accurately established either before or after making the nickel titrations, as the standard of the KCN solution slowly changes with time. By doing so, a correct standard is obtained for the nickel. This is done by taking 3.0 or 4.0 c.c. of the $AgNO_3$ solution, dilute it with 150 c.c. water and add a few drops of the KI solution, then render alkaline with ammonia. Carefully run in KCN solution till the turbidity disappears, add the $AgNO_3$ solution drop by drop from a burette till opalescence appears. As the $AgNO_3$ solution can be made up to contain a known amount of silver, the standardization by this means is accurate ($AgNO_3 \times 0.635 = Ag$) ($Ag \times 0.27193 = Ni$).

Analysis of Commercial Nickel-Copper Matte.

The matte, used for the roasting experiments, was furnished by the Orford Copper Company. It was analyzed and found to contain 55.95 per cent. nickel, 23.50 per cent copper, and 16.19 per cent. sulphur. At first the low sulphur content was questioned, but reanalysis gave the same results. The theoretical percentage of sulphur for the nickel and copper alone, not counting for the 4.36 per cent. of other constituents, is 15.25 per cent. to form Ni_2S, plus 5.88 per cent. so form Cu_2S, a total of 21.13 per cent.

The deficiency of 4.94 per cent. sulphur in the commercial matte, could not be accounted for until a piece was polished, etched with dilute hydrochloric acid (1 part conc. HCl + 4 parts water), and viewed under the microscope, which showed it to be made up of a mixture of Ni_2S, Cu_2S, and metal. Alcohol, containing 2 per cent. by volume of concentrated HNO_3, was tried for etching the specimens, but was found to be not so suitable as an aqueous solution containing 20 per cent. by volume of concentrated HCl. The nitric acid solution attacked the Ni_2S rapidly, and also dissolved the Cu_2S, but more slowly. When the specimens, so etched for two to three minutes, then washed with alcohol and dried, were viewed under the microscope, a cavity was seen where the Ni_2S was,

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the Cu_2S appeared dark and rough, and the metal showed as a bright, yellow-tinged, ground-mass surrounding the Ni_2S and Cu_2S. The etching by means of the hydrochloric acid solution was much more satisfactory, the Ni_2S was slowly dissolved and assumed a dark-brown color after about five to six minutes treatment, whereas the Cu_2S and the metal were not attacked. This solution served the means of distinguishing the constituents of the matte from each other: cavities appeared as black spots, the Ni_2S as dark-brown masses with more or less square corners, the Cu_2S showed as light-blue irregular masses, and the metal as bright yellowish ground-mass surrounding the sulphides. The specimens, after etching, were first washed with water, then alcohol and dried. The micro-photographs show the Ni_2S as dark square-cornered particles, the Cu_2S as lighter and more irregular particles, and the metal as a white ground-mass surrounding the particles.

In order that the sulphur may be proportionately allotted respectively to the nickel and to the copper, a number of analyses of nickel-copper mattes were taken from the article by Mr. D. H. Brown* and the sulphur required to form Ni_2S, Cu_2S, and FeS was calculated by using the factor 0.2725 for the sulphur required for the nickel, the factor 0.250 for that required by the copper, and 0.590 for the iron. These figures were obtained by using the following combining weights: S = 32.0; Ni = 58.7; Cu = 63.6; Fe = 55.9. Table IV. contains the analyses of nine different mattes, and the calculated percentages of sulphur to satisfy Ni_2S, Cu_2S, and FeS. The calculated total S, and the determined total S, are so close in most cases (especially so, in the mattes contain-

TABLE IV.

ANALYSES OF MATTES, AND CALCULATED SULPHUR CONTENT.

Actual Analyses.				Calculated Percentage Sulphur for			
Nickel.	Copper.	Iron.	Total S.	Ni_2S.	Cu_2S.	FeS.	Total S.
10.1	23.1	37.8	26.7	2.7	5.8	22.3	30.8
29.0	14.3	26.3	26.7	7.9	3.5	15.5	26.9
34.0	15.4	21.7	25.9	9.3	3.8	12.8	25.9
37.5	18.9	15.6	26.0	10.2	4.7	9.2	24.1
43.2	20.0	10.3	25.1	11.8	5.0	6.1	22.9
26.1	45.3	5.5	21.2	7.1	11.3	3.2	21.6
24.1	47.8	5.2	22.5	6.5	12.0	3.1	21.6
54.4	20.2	2.8	20.2	14.8	5.0	1.7	21.5
52.3	26.3	0.5	20.1	14.2	6.6	0.3	21.1

* Trans. Am. Inst. Min. Engs., Bulletin No. 40, April, 1910, pp. 302–304.

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ing less than 30 per cent. iron) that we may take for granted that nickel sulphide exists in nickel-copper mattes as Ni_2S and not as NiS, as was formerly claimed.

As the matte used for the roasting experiments contained 16.19 per cent. S, instead of the theoretical amount 21.13 per cent., and it also contained a metallic alloy of nickel and copper, the allotting of the sulphur as Ni_2S and Cu_2S was made in the ratio of the nickel to the copper in the matte, which is as 55.9 : 23.5 :: 70.42 : 29.58. This ratio is the same as the percentage of nickel to copper in "monel metal," and is approximately the same ratio as the molecular ratio ($Ni_5 : Cu_2$), which is 69.76 : 30.26. The sulphides, therefore, exist in the matte in the ratio expressed by the formula ($5Ni_2S \cdot 2Cu_2S$), and the associated metal as an alloy corresponding to the composition of "monel metal."

The calculated composition of the matte, which was roasted, based on this data and these assumptions is as follows:

54.0 per cent. Ni_2S, consisting of 42.4 per cent. Ni, and 11.6 per cent. S.

23.0 per cent. Cu_2S, consisting of 18.4 per cent. Cu, and 4.6 per cent. S.

18.6 per cent. alloy, consisting of 13.5 per cent. Ni, and 5.1 per cent. Cu.

Experimental: Part IV.

Roasting of Commercial Nickel-copper Matte.

The matte used for these experiments was furnished by the Orford Copper Company, and was produced from Sudbury ores. The lump matte was crushed and ground to pass through a 60-mesh screen; it was resifted through an 80-mesh screen, and that portion which passed through the 60-mesh screen and remained on the 80-mesh screen was reserved for the roasting experiments. It was well mixed, sampled and analyzed. It contained 55.95 per cent. Ni, 23.50 per cent. Cu, and 16.19 per cent. S; its calculated composition is given above.

The roasting experiments were conducted with about 300 grams of the matte, by placing it on small roasting pans (with fire-brick bottom) in layers of about three-eighth of an inch. The roasting was conducted in an ordinary assay muffle furnace, fired by gas and blast, and the temperatures determined by means of a LeChate-

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lier thermo-couple placed on the roasting dish, and in circuit with a Keiser & Schmidt galvanometer.

The condition of the roasting was determined by taking samples at intervals of 1 or 2 hours, and making analyses for $NiSO_4$, $CuSO_4$ and total S. The method of analysis was in brief as follows: a 1.0 gram sample was leached with 175 c.c. of boiling water for about 20 minutes which dissolved the sulphates of nickel and copper. The residue was allowed to settle, and the clear solution decanted from the residue. The residue was twice washed by decantation, and the washings added to the main solution. The solution was rendered acid by the addition of about 7 c.c. of conc. H_2SO_4, two strips of metallic aluminium were added (size about $\frac{1}{2}'' \times 1''$), and the solution boiled till all copper precipitated. (Test for presence of copper by adding a small amount of H_2S water.) The solution was allowed to remain undisturbed till clear, and then decanted from the precipitated copper, after which the precipitate was twice washed by decantation with about 10 c.c. water. The precipitate was dissolved in dilute nitric acid and the red nitrous fumes boiled off, then ammonia was added to neutralize the free acid and about 5 c.c. in excess; the solution was cooled, diluted to about 250 c.c. and titrated with standard KCN solution, as outlined under Part III. The nickel solution was prepared for titration by adding 5 grams of NH_4Cl crystals and 8 grams of citric acid, and ammonia added to neutralization point and about 1.0 c.c. in excess; the titration was then made as outlined under Part III. The sulphur content was determined by grinding the sample to pass through 120-mesh screen, and making the analysis as outlined under Part III.

Four roasts were conducted with the following conditions:

Roast No. 1: 10 hours at 430° C. to 450° C.

Roast No. 2: 10 hours at 500° C. to 525° C.

Roast No. 3: 10 hours at 590° C. to 620° C.

Roast No. 4: 7 hours at 675° C. to 700° C.

Rabbling was done at intervals of five to ten minutes.

No sintering of the matte occurred in the case of Roasts no. 1, no. 2 and no. 3, but in the case of Roast no. 4 the material caked considerably and it was necessary to remove it twice from the furnace and break up the lumps.

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The analyses of the samples are given in Tables V., VI., VII. and VIII.

TABLE V.

ROASTING AT TEMPERATURES BETWEEN 430° C. AND 450° C. FOR 10 HOURS.

Sample No.	Time of Roasting, Hours.	Percentage Nickel as $NiSO_4$.	Percentage Copper as $CuSO_4$.	Percentage Total Sulphur.	Percentage Desulphurization.
1	1	0.08	0.0	16.18	0.06
2	3	0.12	0.0	16.05	0.90
3	5	0.28	0.0	16.07	0.80
4	7	0.22	0.0	15.85	2.10
5	9	0.33	0.0	15.74	2.78
6	11	0.14	0.0	15.45	4.57

TABLE VI.

ROASTING AT TEMPERATURES BETWEEN 500° C. AND 525° C. FOR 10 HOURS.

Sample No.	Time of Roasting, Hours.	Percentage Nickel as $NiSO_4$.	Percentage Copper as $CuSO_4$.	Percentage Total Sulphur.	Percentage Desulphurization.
1	2	0.05	0.0	16.10	0.55
2	4	0.14	0.0	16.05	0.90
3	6	0.19	0.0	15.54	4.02
4	9	0.14	0.0	15.43	4.70
5	11	0.09	0.0	15.34	5.25

TABLE VII.

ROASTING AT TEMPERATURES BETWEEN 590° C. AND 620° C. FOR 9 HOURS.

Sample No.	Time of Roasting, Hours.	Percentage Nickel as $NiSO_4$.	Percentage Copper as $CuSO_4$.	Percentage Total Sulphur.	Percentage Desulphurization.
1	1	0.10	0.0	16.20	0.0
2	3	0.19	0.0	15.44	4.63
3	5	0.46	0.0	15.00	7.35
4	8	1.37	0.0	14.20	12.30
5	10	1.56	0.0	13.64	15.75

TABLE VIII.

ROASTING AT TEMPERATURES BETWEEN 675° C. AND 700° C. FOR 6 HOURS.

Sample No.	Time of Roasting, Hours.	Percentage Nickel as $NiSO_4$.	Percentage Copper as $CuSO_4$.	Percentage Total Sulphur.	Percentage Desulphurization.
1	1	0.03	0.0	16.02	1.04
2	2	0.07	0.0	15.35	5.19
3	3	1.36	trace	12.48	22.91
4	5	4.70	0.28	5.36	66.89
5	7	5.80	0.90	2.78	82.82

The figures under the column "Percentage Desulphurization" were obtained by dividing the difference in the percentage total sulphur of the original matte and that of the roasted matte by the percentage of sulphur in the original matte.

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The results of the roasting experiments show that in order to eliminate sulphur from a nickel-copper matte, the roasting must be conducted at temperatures above 600° C., but below 700° C. between which temperatures the desulphurization is rapid and the material is not rendered pasty. Above 700° C., the particles consolidate, and unless vigorous rabbling is continuous, they will cement together (or sinter) as lumps, which retards the desulphurization. The sintering of the roasting material may be prevented by first carrying on the oxidation for about two hours at a temperature approximating 500° C., during which time the particles will have been coated with oxides, then finishing the oxidation at and above 700° C. at which temperatures the roasting proceeds rapidly.

Conclusions.

A summary of the results of the experiments is as follows:

Ferrous sulphate oxidizes to the basic sulphate at temperatures between 280° C. and its decomposition temperature, which is about 580° C. The higher the temperature, the more rapid the oxidation; and also, the more rapid the decomposition the higher the temperature above 580° C.

Basic sulphates of copper are not formed by the direct oxidation of copper sulphate at temperatures below its decomposition temperature, which is about 647° C., where the decomposition begins. Above this, the breaking down of the sulphate occurs more rapidly the higher the temperature.

Basic sulphates of nickel are not formed by the direct oxidation of dehydrated nickel sulphate, but form at temperatures above its decomposition temperature. Nickel sulphate begins to decompose at 727° C., and takes place rapidly above 736° C.

Cuprous sulphide freezes at 1110° C.

Nickel sulphide, of the composition corresponding to Ni_5S_4, freezes at 790° C.

A mixture of 64 parts of Cu_2S and 36 parts of Ni_5S_4 begins to solidify at 948° C., where the excess Cu_2S separates, after which the eutectic freezes at about 740° C.

A mixture of 50 parts of Cu_2S and 50 parts of Ni_5S_4 begins to freeze at 900° C., where the excess of Cu_2S solidifies, and at about 740° C. the eutectic freezes.

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A mixture of 40 parts of Cu_2S and 60 parts of Ni_5S_4 is the eutectric; it freezes at about 740° C.

Mattes, produced by smelting nickel-copper ores, contain the nickel in the form of the subsulphide Ni_2S. These mattes, when "blown" to almost complete elimination of the iron sulphide, contain an alloy of nickel and copper of composition of these two metals in the proportion in which they existed in the matte.

The most suitable conditions for the rapid roasting of nickel-copper matte are: start the roasting at temperature between 450° C. and 500° C., and hold at this for sufficient time so as to coat the particles with a film of oxides, after which continue the roasting at temperatures between 675° C. and 750° C. until the desired desulphurization has been accomplished. Continuous rabbling necessary to prevent sintering.

(Experiments on the roasting of nickel-copper matte, and the reaction of the sulphides and oxides of nickel and copper, have been conducted subsequent to the experiments included in this article. It is expected that these later experiments will be the basis of an article to appear in a future number of the QUARTERLY.)

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ALUMNI NOTES.

At a joint meeting of the Board of Directors and of the Council of the American Institute of Mining Engineers, Dr. R. W. Raymond tendered his resignation as secretary. He has been relieved from the many executive and administrative duties which have been increasing steadily, and which with the suggested extensions of the activity of the Institute must further multiply. Dr. Raymond has been appointed secretary emeritus of the Council, the Institute retaining his services for special editorial and other duties for which he is so well qualified by training and experience. Dr. Raymond, one of the founders of the Institute in 1871, was elected vice-president and became president in that year, following David Thomas, who resigned because his advanced years did not enable him to perform the duties of his office to his own satisfaction. He was annually reëlected president in 1872, 1873 and 1874. In 1883 he was appointed secretary to succeed Dr. T. M. Drown, who had resigned. He was elected secretary in February, 1884, and has been annually reëlected since then.

Dr. Joseph Struthers, '85 Mines, who has been connected with the Institute as assistant secretary for eight years, and as editor for five of those years, has been elected secretary of the board of directors and has been appointed secretary of the council. Dr. Struthers was editor of the Mineral Industry for the years 1900–1903 (Vols. 8, 9, 10 and 11), and prior to that time was for fifteen years on the teaching staff of the department of metallurgy of the School of Mines, Columbia University. He has been field assistant of the U. S. Geological Survey, in charge of the preparation of numerous reports for the mineral resources, and special agent of the U. S. Census in a like capacity. In connection with the work of the Institute, he is a member and treasurer of the board of the United Engineering Society, chairman of its house committee, and secretary of the library conference committee, which has general supervision of the libraries of the engineering societies in the United Engineering Society Building, 29 West 39th Street, New York City.

J. Maxwell Carrere, C.E., '83 School of Mines, general manager of the Blanc Stainless Cement Co., Allentown, Pa., delivered an address to the Canadian Cement and Concrete Association on March 7th.

"I wonder how many of you stop to think of the man, his struggles, anxieties and years of toil for your welfare—when in the course of a business day—as a matter of fact, you open your mail—brought to you by the locomotive or the steamer—when you receive a telegram, and immediately reach for your phone—and rushing for the elevator descend to the street—make for the subway, and being held up by excavations you mentally, if not actually, rail at the contractor and his men handling that nasty, sloppy concoction you know—is concrete?

"What would have happened one hundred years ago to you under the

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same conditions—granted that a letter was mailed only a few days ago from the mother country or the province of Louisiana? Think that over!

"*Historical.*—All events follow the course of a nation—and necessity creates achievements.

"With the rapid growth of our country in the last century, material development predominated, and the intellectual could not keep pace with the times, excepting in those communities—well-populated from the first and located at the seaboard—where the continual influx from the old country kept pouring into the states in ever-increasing numbers, and bringing with them—if not new ideas—at least new problems to be solved by the young nation. Problems of government, religion, education, hygiene—all subjects that required, not only the practical, but the learned man of these communities in their solution.

"Our institutions of learning were few, and while endowed with enthusiasm of the learned early settler, were struggling under great difficulties, supported by the devoted love and loyalty of their faculties and early graduates.

"The material development of the country had created great individual wealth, and with this wealth came the desire of the individual to make his progeny the greatest ever. And having found in the course of his own experience that whenever a knotty technical problem was involved, the foreign engineer or professional man had solved it for him, it is most natural that his first idea was to send his sons abroad for that education he did not have himself. Later, through national pride, he wisely spent that very wealth to endow our institutions of learning that they might even surpass similar institutions abroad, or at least to furnish that particular education he conceived young America needed. And thus began the rapid development and enlargement of our colleges, and towards the end of the nineteenth century with rapid strides we see colleges springing up in every direction, and the older colleges becoming universities.

"With this period in our national history we also perceive the renaissance of art in America—and I say renaissance wittingly—for the Indian and the early settler had a conception of art. Building operations had increased at a tremendous pace, the architect was coming into his own—he was no longer a contractor, but a recognized authority. The engineer already recognized—seemed to develop a desire to create the impossible. To span rivers he had to invent the steel cable, to take care of increased traffic he had to design the elevated roads, and when that traffic again increased he had to create the tube. To do all of which he no longer could rely on the Roman and natural cements, but had to create that product which made those structures possible—Portland cement.

"Portland cement is the best product of its kind devised before the twentieth century. It superseded the Roman, the Puzzolans and the Grappiers, and later, the natural cements, because of its more perfect chemical structure, physical strength and economic value. It is the cement of today. The cements of yesterday have passed into history.

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"Tonight, on behalf of the men engaged in our industry, it is my purpose to give you an account of our stewardship, and point out to you in what manner we hope to serve the present and future generation.

"Portland cement, like its predecessors, is a foreign discovery or product which has been adapted and developed by American engineers. Twenty years ago the production of American Portland cement was small and the methods of production primitive—being copied from foreign countries where labor was cheap and fuel high; whereas the reverse conditions existed in the United States and England. An English engineer, to save labor, conceived the rotary kiln—a revolving cylinder in which to burn the product and save labor. American ingenuity perfected it, and further improved it by perfecting a method of burning powdered coal. To two engineers—one American and one an Englishman—Messrs. Seamans and Hurry—backed by strong financial assistance, do we owe the first real development that put Portland cement within reach of every man. That they have fared well can but be gratifying to any man who admires achievement, and in fact I will go further and state that the fundamental principles of the cement mill as laid down in the first plant designed by them at Northampton, Pa., remained such without alterations or improvements of a vital nature till a few years ago, when the wizard of Menlo revolutionized the industry with his long kiln and grinding process. While both were decried by everyone at first, the long kiln has been universally adopted both here and abroad—with the reservation often expressed that there is nothing in it—but volume.

"The future will demonstrate the value of Mr. Edison's other conceptions. For my part I consider most of them admirable, but an engineer must always consider the economic side of the problem as well, and not having had the opportunity of doing so on his other conceptions, I can only say that they are fundamentally sound, and that the industry today is in such a state of perfection that past great economies are no longer possible.

"During all this period of less than twenty years building operations increased at a tremendous pace and Portland cement became the recognized structural cement. It is the cement of today. It is a grayish, sometimes a bluish dull material of great structural value for the engineer, the architect and the builder, but its appearance precludes the possibility of its ever attracting the artist. There is hope, therefore, that the cement of today—the gray Portland cement—will remain the structural factor in the future. But the cement of tomorrow is the *white Portland cement.*

"The *white* Portland is purely American, for while scientists knew that a white Portland cement could be produced from pure kaolins and pure limestone—and some companies tried it—the cost of such a product was commercially impossible. Practically, I consider it equally unattractive. After years of toil in experimenting—four of which were spent in the present plant—Allentown, Pa., gave birth to the *white Portland cement* of commerce.

"The history of four years since the *white* Portland cement was

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introduced is most interesting, for they were years of distress and panic, years of curtailment of operations, years of economy. They are interesting because of the possibilities created for the gray by the *white* Portland cement, because of the artistic possibilities in a cement structure of tomorrow, and because of the further artistic education it may create for the welfare of the general public.

"In quality there is no difference between the gray and the white. In purity, color and adaptability, the *white* is the darling. Like in many other paths of life, there is no occasion for jealousy, for being of one nature, one quality, one family, they can be made to harmonize ideally in attractive and durable construction, heretofore unattainable.

"The *white* Portland cement destroyed the distinction between structural and ornamental when applied to cement—it no longer exists. The engineer can build his concrete bridge or his stack as artistically as he can conceive it of Portland cement, and has no possible excuse for the creation of an unattractive structure.

"Portland cement is plastic—*white* Portland cement is not only plastic, but admirably adapted for ornamental work, no matter how elaborate—and besides can be colored any tint to suit the particular artistic temperament of the individual, and for this reason I feel that this baby industry will not only greatly benefit the gray Portland cement industry, but the individual and the public in general by putting within reach of all the attractive home and pleasant surroundings constructed throughout of indestructible material—Portland cement.

"Our public today craves for the refined and the beautiful, and while in our youth as a nation we still demand the big, and sometimes the showy display of our conceptions, yet education, travel and association have raised the public standard in general to an appreciation of the fitness of things, to the difference between large and great, quantity and quality, and also to an appreciation that 'in union there is strength' for the large and the great, the quantity and the quality, the gray and the white, can be united to better serve our ends if properly assimilated."

By lantern slides he illustrated some of the work accomplished by the architect and the engineer, and showed that in the union of the white and the gray there is not only strength, but also character.

EX LIBRIS.

In the Columbia University Library is an interesting collection of book-plates, many of which were the property of students, alumni and officers of the University. The collection has been accumulating for many years, and is now being put into convenient shape for preservation and consultation. Each book-plate will be mounted upon a separate sheet, and properly catalogued. Eventually, the collection will be a valuable memorial of men and women in any way connected with Columbia University, whose love of books led them to adopt a distinctive book mark. It will form an interesting addition to the collection of Columbiana, which is intended to include all material in manuscript or in print directly or indirectly relating to the University.

The book-plate collection is now very incomplete, but a systematic

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effort is being made to supply its deficiencies. Students, alumni, and officers are therefore asked to send to the Librarian of the University examples of their personal book-plates, and any other Ex Libris which they are willing to have preserved in the Library.

The Iron Silver Mining Company, of Leadville, Colorado, is managed by George O. Argall, '04S.

The firm of Dier and Bates, civil and mining engineers, Bozeman, Montana, of which William S. Bates, A. A. Dier, and George O. Arnold '07S are members, recently undertook the construction of the New Gallatin Electric Railway.

At Guayaquil, Ecuador, South America, Luis Alberto Carbo, '04S, is chief engineer of the Sewer and Water Board, also of the High Pressure Fire Service Plant.

Professor Charles Derieth, Jr., '96S, has been elected president of the San Francisco Association of members of the American Society of Civil Engineers. He is dean of the College of Civil Engineering, University of California.

Maurice Deutsch, '06S, who has recently contributed important papers to these columns and has been with the Foundation Company for the past five years, is now associated with Mr. Henry G. Opdycke in the practice of civil and consulting engineering, at 30 Church St., specializing in the design of difficult foundation work and mining shafts to be sunk through deep water bearing strata.

Benjamin B. Gottsberger, '95S, is general manager of the Miami Copper Company at Miami, Arizona.

Frederick R. Hutton, '73, '76S, Emeritus Professor of Mechanical Engineering, has been appointed Consulting Mechanical Engineer to the Department of Water, Gas, and Electricity of New York City.

Professor James F. Kemp, '84S, has been elected a member of the National Academy of Sciences.

Daniel E. Moran, '84S, has retired from active participation in the management of the Foundation Company to engage in the practice of engineering with offices at 55 Liberty Street, New York. He remains Consulting Engineer for the company.

Benton Overman, '09S, who went to Africa after graduation, is now engaged in mining engineering at Fimiston, West Australia.

Sheldon Smillie, '04S, is chief engineer of the Quincy Mining Company, Hancock, Michigan.

August Ulmann, Jr., '06, '08S, is constructing a large rope factory at Matanzas, Cuba, for Raffloer, Erbsloh and Company.

J. R. Wemlinger, '00S, president of the Wemlinger Steel Piling Company, New York, has been elected secretary of the American Society of Engineering Contractors.

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ABSTRACTS—ANALYTICAL CHEMISTRY.

By E. WALLER.

Device in Distilling. Downs (*J. Ind. Eng. Chem.*, III., 110). When distilling gas tars in a side-neck flask, the samples frequently contain so much water that the mass will "bump" violently and often particles of tar are thrown in such a way as to go into the condenser. The trouble may be remedied by hanging two concentric cylinders of brass wire gauze (30 mesh) in the neck of the flask, supporting them by wires held in place by the cork. Plenty of room for the thermometer bulb can be left inside the inner cylinder.

Sensitive Indicator. Sacher (*Chem. Ztg.*, XXXIV., 1192). An alcoholic extract of radish skins (96 per cent. alcohol) affords a very sensitive indicator, red with acids, green with alkalies. Like phenolphthalein it is affected by CO_2.

Potassium as Platinchloride. Fresenius and Brinton (*Zts. anal. chem.*, L., 21). Investigations have been made as to the effect of the presence of Ca and Mg chlorides in this estimation. It was found that the presence of $CaCl_2$ caused an overestimation of K by 0.5 per cent. With $MgCl_2$ the error was only 0.3 per cent. The conventional 80 per cent. alcohol was found to be the best strength to use. The use of 70 per cent. alcohol reduced the results by nearly 1 per cent.

Strontium and Calcium—Qualitative. Hinds (*Chem. News*, CIII., 157). The author shows, by the record of experiments on solutions of known strengths, that the sensitiveness of Ca and Sr ions to $(NH_4)_2C_2O_4$ is about the same, but that with Na_2SO_3 the $SrSO_3$ is the more insoluble—prone to precipitate—which difference is more marked when the solution is acidified with acetic. On these facts, he has worked out a plan for qualitative detection of either one of these elements (ions) in the presence of the other—applicable to most cases. When the dilution is very considerable so as to give room for uncertainty, the plan recommended is to use two Pt wires, the one dipped in a $CaCl_2$ solution, the other in the solution to be tested. Holding both parallel in the same hand and drawing them simultaneously through the Bunsen flame, the comparison of flame colorations is usually such as to indicate clearly whether Sr is present or not.

Calcite and Aragonite. Thugut (*Chem. Centr.*, II., 1084). Congo red or an alkaline solution of alizarin will stain aragonite pink, but do not color calcite. A still more sensitive reagent is $AgCrO_4$. Treat the mineral, coarsely powdered with $N/10AgNO_3$ for a second, wash off and then moisten with a 20 per cent. $K_2Cr_2O_7$ solution. Aragonite stains a deep red, but calcite shows no color.

Barium—Qualitative. Curtman and Frankel (*J. Am. Chem. Soc.*, XXXIII., 724). In the qualitative examination of 1 gm. of material containing members of the other groups, it was found that 25 to 50

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mgms. Ba might escape detection. A study of the causes has been made. It is found that oxidation of the sulfids in the H_2S group, or oxidation due to $FeCl_3$ cause losses as $BaSO_4$. Also absorption of CO_2, when precipitating with NH_4OH, as well as oxidation of the sulfides in separating the $(NH_4)_2S$ group, and finally $(NH_4)_2CO_3$ will fail to detect 10 mg. Ba in the presence of much NH_4 salt. Aside from these points, many of the reagents contain small amounts of sulfates, which remove the Ba during various stages of the examination.

Barium—Separation from Calcium and Magnesium. Gooch and Boynton (*Am. J. Sci.*, XXXI., 212). In concentrated solutions, acetyl chloride precipitates $BaCl_2 2H_2O$, but not $CaCl_2$ or $MgCl_2$. In applying the process use a mixture of acetone with acetyl chloride (4:1). The solution of the chlorides should be at 15° C. and contain about 0.1 gm. $BaCl_2 \cdot 2H_2O$ per c.c. To this about 30 c.c. of the mixture should be added at the rate of 5 drops per second with constant shaking. The separated $BaCl_2$ may be filtered on a Gooch, washed with acetone and weighed up if desired. Separation by this means from $SrCl_2$ is not very satisfactory.

Manganese—Volhard Method. Cohen and Little (*Analyst*, XXXVI., 52). Volhard's method consists in titrating hot with permanganate, a sulfate solution over-neutralized with ZnO, and containing $ZnSO_4$. The results are usually a little low, and the presence of chlorides, as well as the temperature were found to have some effect. Fischer's modification consists in neutralizing almost exactly with NaOH, then adding 1 gm. of freshly ignited ZnO and 10 gm. $ZnSO_4$, heating to boiling and titrating at as near boiling as possible, boiling and shaking frequently until a permanent pink is obtained, then adding 1 c.c. glacial acetic which causes decolorization as well as subsidence of the precipitate and then cautiously adding a few drops of the permanganate at a time until the pink persists.

The authors find that 5 gm. of $ZnSO_4$ is sufficient; that the titration can be accurately accomplished in the presence of 10 to 40 gm. NaCl, and that the solution, though it should be hot, should not be boiled after the addition of the acetic.

Nickel and Cobalt. New Qualitative Test. Weil (Bull. Soc. Chem., IX., 20). In neutral solutions K_2CrO_4 produces precipitates with these metals: $NiO \cdot NiCrO_4$, chocolate brown, forming slowly in the cold, readily on boiling; and $CoO \cdot CoCrO_4$, reddish brown, adheres strongly to the glass, forms readily in the cold except when solutions are dilute. In that case boiling effects precipitation. These compounds are soluble in weak acids and in NH_4OH. When both metals are present in approximately equal proportions, adding a 10 per cent. solution of K_2CrO_4 precipitates Co in the cold, on filtering and boiling the Ni is precipitated. With much Co and little Ni this procedure can be followed and if the precipitate obtained by boiling is washed and then dissolved in NH_4OH, on evaporating slowly the Ni compound separates with its characteristic color, whereas the Co compound shows dirty green. With much Ni and little Co the Co compound does not separate in the cold, but the treatment described above with NH_4OH will bring out the indication of the presence of Co.

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Zinc—Separation and Determination. Piperant and Vila (*Proc. Seventh Int. Cong. Appl. Chem.,* London, 1909, Sect. I., 141). The method depends on boiling a caustic alkali solution of ZnO with pulverized S, which precipitates ZnS. When S is in excess the ZnS shows a pink tint. For ores, decompose 5 gm. with aqua regia, and filter off the insoluble portion, running the solution and washings into a 250 c.c. flask. Then add NaOH solution in sufficient amount to redissolve all Zn, fill to the mark, mix thoroughly by shaking and filter through asbestos. Fifty c.c. of the clear filtrate are then much diluted and heated to boiling in a 500 c.c. Erlenmeyer. To the boiling liquid add 2 to 3 gm. finely pulverized S. Continue the boiling, which brings down all Zn as ZnS. Fill up to about 400 c.c. with boiling water and let stand about fifteen minutes. Wash by decantation, finishing by pouring hot solution of $(NH_4)_2S$ over the precipitate. Roasting to ZnO in a muffle and weighing up as such is recommended.

With lithopone, agitate with water and Br in the cold, heating up afterward, and eventually digest with aqua regia. The other details are the same as above indicated.

Iron, Aluminum, Chromium—Separation. Tcharviani and Wunder (*Ann. Chim. Analyst,* XVI., 7). Fuse the mixture of oxides for several hours with Na_2CO_3. Treat the melt with water, which leaves only Fe_2O_3 undissolved. Adding an excess of NH_4NO_3 will then precipitate Al_2O_3 in purer condition than by the usual methods. Acidify the filtrate with HNO_3, reduce with alcohol and precipitate Cr_2O_3 with NH_4OH. Redissolve and reprecipitate to obtain it pure.

Chromium in Chrome Tungsten Steel. Wdowiszewski and Bogalubuff (*Chem. Ztg.,* XXXIV., 1365). In a half liter Erlenmeyer flask treat 1 to 2 gm. of the borings with 10 or 15 c.c. of 15 per cent. Na_2HPO_4 solution per gm. and 7 to 8 c.c. H_2SO_4 (gr. 1.65) per gm. of borings. Finally add 5 c.c. of water and warm. Decomposition results in about 15 minutes. Add 2 c.c. HNO_3 and increase the heat to dissolve the W. Dilute to 300 to 500 c.c., add 3 to 5 gm. $(NH_4)_2S_2O_7$ and boil until the persulfate is all decomposed. Cool, add a measured amount of standard $FeSO_4$ and titrate the excess with standard permanganate.

Aluminum Ores. Calafat y Leon (*Anal. Fis. Quin.,* VIII., 281). Fuse about 0.5 gm. of the mineral with an equal volume of dry Na_2CO_3 for some minutes, extract with water and pass CO_2. If the material is of value as a source of Al, material amounts of $Al_2(OH)_6$ quickly precipitate out.

Alumina—Precipitation in Granular Form. Taylor (*Ch. News,* CIII., 169). Instead of boiling when adding NH_4OH, bring the solution to the temperature of 66° C., inserting a thermometer to indicate that point, then add NH_4OH, stir in and boil. Under these conditions the precipitate takes a granular form and is readily filtered and washed.

Free Acid and Basic Alumina in Aluminum Sulfates. Craig (*J. S. & C. I.,* XXX., 184). The method depends on the fact that KF in excess produces in Al salts stable salts which have no effect on phenolphthalein indicator, leaving free acid in condition to be accurately

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titrated. Thus: $Al_2(SO_4)_3 + (xH_2SO_4) + 12KF = 2(AlF_3 \cdot 3KF) + 3K_2SO_4 + (xH_2SO_4)$.

The solutions required are: Phenolphthalein 0.2 per cent.

KF solution sp. gr. 1.35, approximately 0.5 gm. per c.c. The solution is made up at first a little stronger than this, and then exactly neutralized (with KOH or H_2SO_4 as required), diluted to the above strength and filtered clear. It should be preserved in a bottle internally coated with wax.

Standard H_2SO_4, preferably *N/2*, (may also be normal or decinormal).

Standard KOH, to correspond.

These last should be clear and free from Al_2O_3 or other bases of that class.

A weighed portion of the sample should be dissolved in such proportion as to give 1 to 3 gm. Al_2O_3 in 200 c.c. This is filtered and 20 c.c. portions are used for testing. This 20 c.c. is gradually added with stirring to a mixture of 10 c.c. KF solution with 50 or 60 c.c. of water and 0.5 c.c. phenolphthalein.

If free acid is present the solution remains colorless and it should be titrated with the standard KOH to a permanent pink.

If basic Al_2O_3 is present the reaction is of this type: $Al_2O(SO_4)_3 + 12KF + H_2O = 2(AlF_3 \cdot 3KF) + 2K_2SO_4 + 2KOH$. Titrating back with standard H_2SO_4 will give only an approximate measure of the basicity, so a fresh 20 c.c. of the solution is treated with the amount of standard acid indicated plus 4 c.c. or so more, the mixture is heated, and after cooling, the KF solution and phenolphthalein are added as before, and the acidity determined. By subtraction the basicity is found in terms of *N/2* solution. One c.c. *N/2* = 0.0085167 gm. Al_2O_3. The effect of Na salts, if present, should be prevented by addition of 20 to 30 c.c. of neutral saturated solution of K_2SO_4. NH_3, if present, should be expelled by heating with a known amount of KOH solution before making the test.

Lead in Tin Alloys. Schacherl (*Arch. Chem. Micros.*, III., 45). One to 1.5 gm. of the sample are treated with 2 or 3 c.c. of a mixture of 2 parts HBr (gr. 1.49) with 1 part Br in a 200-c.c. flask. Cool down if the action is too violent. Warm on the water bath to complete solution, adding a little more of the Br mixture is necessary. Add about 10 c.c. of water and then, a little at a time, a 20 per cent. solution of Na_2S until the precipitate is clear black. A persistent brownish color in the precipitate would indicate SnS, which is remedied by adding a little S. Dilute largely and let stand over night. Decant the clear solution and digest the precipitate with 20 c.c. Na_2S solution. Filter and in the precipitate determine the Pb.

Cuprous Oxide in Metallic Copper. Coffetti (*Gaz. chem. Ital.*, XXXIX., 137). In the absence of air, Cu_2O is soluble in NH_4OH. A special form of apparatus is used consisting of two separators mounted one above the other, the lower having a side inlet tube below and a side exit tube above, independent of the stop cock and tubulure, so that a continuous stream of H gas can be passed through the lower separator

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during a test. The Cu filings, cleaned and dried, are placed in the lower separator on a layer of glass wool. The H stream is then passed through strong NH_4OH to carry NH_3 to the Cu. After a short time, the Cu is washed repeatedly with NH_4OH of 10 per cent. strength run in from the upper separator, which is drawn off below, until the washings are colorless. Fifty c.c. concentrated NH_4OH are then run in and allowed to stand on the sample for 4 or 5 hours. This is run off and washing again repeated until they show no Cu. The Cu is then determined in the combined NH_4OH solutions and washings and calculated to Cu_2O.

Tin Ores—Decomposition. Loram (*Proc. Chem. Soc.*, XXVII., 60). The ore should be ground to 60-mesh size and then fused in Ag crucible, the charge being essentially in the proportions of: ore 1 gm., KOH or NaOH 6 to 7 gm., KCN 1 gm. Fuse for about an hour. After cooling, the whole readily dissolves in hot diluted HCl.

Tin Stone. Gilbert (*Zts. off. chem.*, XVI., 441). The decomposition by fusion with NaOH is a matter of some difficulty, but if about 2 per cent. of powdered charcoal is mixed with the ore, the result is easily obtained. So small an amount of C seems necessary, that the author considers that it has a catalytic action.

Tin Alloys. Kietreiber (*Oester. Chem. Ztg.*, XIII., 185). Heat 1 gm. with 10 to 15 c.c. HNO_3 until all action is over. Transfer to a capsule and evaporate down with 25 gm. crystallized $Na_2S \cdot 9H_2O$ and heat until the mass begins to harden. Then digest with hot water. Sn and Sb are dissolved, leaving PbS and CuS insoluble.

Tin and Antimony in Solder. Goodwin (*J. Ind. Eng. Chem.*, III., 42). The filings should be fine enough to pass a 30-mesh sieve. For Sb, introduce 2 gm. into a 300 c.c. Jena Erlenmeyer flask, add 5 gm. solid $KHSO_4$ and 10 c.c. concentrated H_2SO_4. By use of test tube holder or some similar device, manipulate the flask over a naked flame until most of the free acid is expelled and no S remains. Do not attempt to heat to dryness. Stand on a sheet of asbestos to cool, and when it is safe to do so add 25 c.c. water and 5 c.c. HCl. Heat again to complete solution and expel any SO_2. The Sn is now stannic, the Sb as antimonous. Cool, add 100 c.c. of cold water and titrate rapidly with *N*/20 permanganate. The permanganate is standardized by use of C.P. sodium oxalate (Sorensen's). For Sn place 0.2 gm. in a flask as before, add 5 c.c. of 15 per cent. Na_2CO_3, then 20 c.c. hot water, then 25 c.c. concentrated HCl. Add one drop of 5 per cent. $SbCl_3$ solution. Close the flask with a cork carrying a U tube of 1 mm. bore, the free end of the tube extending nearly to the level of the bottom of the flask. Heat to a gentle boil on the hot plate until the alloy is decomposed, leaving a small residue of black Sb. Then, without slackening the heat, bring a test tube of 15 per cent. Na_2CO_3 under the free end of the U tube. Cool the flask finally under the tap, allowing the solution to suck back into the flask. When cold add 5 c.c. of 15 per cent. Na_2CO_3 and 5 c.c. of starch solution and titrate with *N*/20 I solution, which has been standardized on pure Sn treated in the same manner.

Antimony—Colorimetric. Schidrowitz and Goldsbrough (*Analyst,*

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XXXVI., 101). For determination of small amounts the coloration produced by passing in H_2S can be used as a means of estimation, solution of gum acacia being used to cause the precipitate to assume the colloidal condition. If the solution is hot when the H_2S is passed in, the colors will be deeper in tint. Ordinarily the use of solutions at ordinary temperatures will suffice (down to 0.02 mgm. Sb). The standards are made up by dissolving weighed amounts of tartar emetic. Comparisons are made in bulks of 10 c.c., to which in every case is added 0.1 c.c. of a 10 per cent. solution of gum acacia. The solutions are then saturated with H_2S. Of course, other members of the H_2S group interfere. Organic matter does also. In case of the presence of such substances, the Sb may be precipitated out by heating with metallic Cu, and the Sb dissolved off of it by warming with KOH and a little permanganate. After filtering from the Cu, etc., the excess of permanganate can be destroyed by tartaric and aliquot portions of the solution submitted to the test.

Gold and Silver in Rich Copper Ores. Lövy (*Chem. Ztg.*, XXXV., 278). Sand should be added in amount sufficient to bring the Cu down to at least 6 per cent. Then on making the ordinary crucible fusion, a Pb button is obtained which can readily be cupelled in the ordinary manner.

Cyanide Solutions for Gold and Silver. Holt (*Min. and Sci. Press*, C., 863). Boil 5 to 10 A.T. of the solution, adding 15 to 20 c.c. saturated solution of PbA_2, 10 to 15 c.c. HCl, and a 1½ inch square of Al foil. Heat until all Pb is deposited on the Al. Then take out the sponge and squeeze it as dry as may be, first by pressure with a rod, later by rolling it out with a round bottle on a square sheet of Pb (2½ inch). Fold up the latter, sponge and all, and drop into a hot cupel.

Wet Gold Assay. De Luce (*Min. and Sci. Press*, C., 895). Prepare a solution containing per 100 c.c. 2 gm. I and 4 gm. KI. Use 50 c.c. on one A.T. of the sample, grinding them together for 10 minutes and then allowing to stand for an hour. If the I coloration disappears, add in more of the I solution. Filter and wash, then shake the solution with a few gm. of Hg in a glass-stoppered bottle. From the amalgam dissolve out Hg by HNO_3, wash the residual Au, dry, ignite and weigh. Hawson (*ib.*, C., 936) finds that with raw sulfides or with telluride ores this mode of extraction is incomplete. Roasting and then applying the treatment gave better results.

Platinum Metals. Glow Reaction. Curtmann and Rothberg (*J. Am. Chem. Soc.*, XXXIII., 718). The absorption of H, illuminating gas, etc., by spongy platinum, as illustrated in the well-known Döbereiner lamp, is the basis of this method of testing. The test is applied by soaking up 0.2 c.c. of the solution to be tested in a bit of thin asbestos paper, then inserting the asbestos in the flame until it is red hot. It is then withdrawn and while still hot is held in the stream of air and gas of an unlighted Bunsen burner. If Pt or certain others of that group are present, the asbestos shows a glow lasting some time. The results of the examination of this test are:

1. That it is applicable for detection of Pt, Pd, Rh or Ir.

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2. The limits of sensitiveness are: Pt 0.002 mg., Pd 0.0005 mg., Rh 0.0009 mg. and Ir 0.005 mg.

3. The presence of other substances practically does not interfere with the test.

4. The test serves as a ready method of testing for the presence of members of the group, Ru and Os excepted. In most work it might be regarded as distinctively a test for Pt.

Separating Platinum from Tin. Wöhler and Sprengel (*Zts. anal. Chem.*, L., 165). The separation of these elements is usually extremely difficult. The authors find the best method to be boiling with H_2SO_4 to precipitate hydrated SnO_2, and then electrolyzing the solution containing the Sn compound in suspension. A current of 1.1 to 1.7 volts and ND100 = 0.2 amp. was used successfully. The suspended SnO_2 is allowed to settle, and is only stirred up toward the end of the operation.

Uranium—Volumetric. Ibbotson and Clark (*Chem. News*, CIII., 146). U solutions are rapidly reduced when hot, but they reoxidize too rapidly to permit of accurate titration. In the cold they reduce slowly and with difficulty and reoxidize slowly under the influence of the air. The method proposed is to heat the solution of U containing 2 to 5 per cent. free H_2SO_4 in a flask containing about 50 gm. pure amalgamated Zn (20 to 30 mesh) until a dark brown color is developed. Then cool quickly under a stream of water, and when cold filter through a small pulp filter. Aspirate air through the filtrate for a minute, and then titrate with permanganate. If Fe is present, that element can be reduced by SO_2 for titration as the U is not reduced by SO_2.

Ferro-Uranium. Trautmann (*Zts. angew. Chem.*, XXIV., 61). Dissolve about 0.5 gm. in aqua regia (this may require 3 hours). Dilute, and without filtering drop in a large lump of $(NH_4)_2CO_3$. Stir frequently during 24 hours, then filter, and boil the filtrate for some time, which precipitates most of the U. The solution is by this time slightly acid—addition of a few drops of NH_4OH will precipitate the rest of the U. Filter and wash with weakly ammoniacal NH_4NO_3. Ignite in porcelain and weigh U_3O_8. C is determined by combustion in O, Si by dissolving in aqua regia and evaporating repeatedly with HCl. Al is frequently present. Fuse the powdered sample with Na_2O_2, extract with water, and precipitate SiO_2 and Al_2O_3 by NH_4OH. In the ignited precipitate SiO_2 can be removed by HF treatment, or fusion with $KHSO_4$ may be used.

Tungsten Determination. Mdivani (*Bull. Soc. Chim. Belg.*, XXV., 41). $SnCl_2$ is stated to precipitate W completely as W_2O_5, which when washed, dried and ignited is converted to WO_3, which is free from Sn.

Euxenite Analysis. Hofmann (*Ber.*, XLIII., 2631). The original paper should be consulted. The mineral contained scarcely any SiO_2, the formula corresponding to $3TiO_2 \cdot 2RO$. Zr, Nb, Er and other rare earths as well as U were present.

Titanium—Colorimetric. Gautier (*Rev. gen. Chim.*, XIV., 16). In the colorimetric estimation of TiO_2 by means of H_2O_2, it is proposed

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to use methyl orange solution for color comparisons. 1 gm. is dissolved in 500 c.c., and of this solution 10 c.c. are diluted to 200 c.c. Of this dilution 0.3 c.c. = 2 per cent. Ti, 0.8 c.c. = 4 per cent. and 1.6 c.c. = 6 per cent.

Titanium Determinations. Bornemann and Schirmeister (*Metallurgie,* VII., 71). Precipitation by NH_4OH is preferable when it can be applied. If the solutions contain fixed alkalies some will adhere to the precipitate in a form that cannot be washed out, giving results too high. On the other hand some Ti nitride may exist in the precipitate which gives results too low. On ignition the precipitate shows a brownish color. The nitride can be removed by evaporating the unignited precipitate with H_2SO_4, containing a little HNO_3. This treatment is ineffective if the precipitate has been ignited. A subsequent washing with water will remove the alkali as sulfate. For final weighing, the TiO_2 should be ignited at 1000° C. for fifteen minutes.

Titanic Acid. Bornemann and Schirmeister (*Metallurgie,* VIII., 646). TiO_2 which has been ignited below 700° is more soluble in conc. H_2SO_4 than when ignited at above that temperature. After fusion with $KHSO_4$ or Na_2CO_3 one can obtain a practically clear solution not precipitable on boiling by use of H_2SO_4 diluted in the proportion of one part to two parts of water. When precipitated by NH_4OH, the undried precipitate is soluble even in SO_3 water. Solutions in strong H_2SO_4 are not absolutely clear if heated to boiling.

Separating Tantalum and Columbium. Foote and Langley (*Am. J. Sci.,* XXX., 401). The method of Weiss and Landecker (vid. QUARTERLY, XXXI., 272) by passing CO_2 which is stated to precipitate Ta only, is found to be ineffective, the Nb being co-precipitated.

Monazite. Wells (*Mineral Resources of the U. S.,* 1909,—*Monazite and Zircon,* U. S. Geol. Survey). It is exceedingly difficult to get a complete decomposition by any single operation. Fusion with Na_2CO_3 and extracting with water was quite as effective as the usual acid treatment, and had an advantage when phosphates were to be determined. Three or four treatments are usually necessary. All the Th is precipitated by $H_2C_2O_4$. The H_2O_2 method for ThO_2 is a quick method but its accuracy depends on a compensation of errors. Th is not completely precipitated, and it appears practically impossible to obtain a precipitate completely free from Ce. Even with the $Na_2S_2O_3$ method, the precipitation must be repeated to get rid of Ce.

Tellurium. Rosenheim and Weinheber (*Zts. anorg. Chem.,* LXIX., 266). Precipitation by hydrazine compounds is not complete or satisfactory, even on boiling a long time. Heating in an autoclave under pressure at 130° for 30 minutes is effective, and the Te is more compact. Telluric acid cannot be titrated directly, but with glycerol it forms a compound acting as a monobasic acid (phenolphthalein indicator). Results may be 3 per cent. too high or too low. The best method is to precipitate as $BaTeO_4$ by use of a known amount of $N/10NaOH$ containing $BaCl_2$, and then titrating back the excess of NaOH.

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Perchlorate in Potassium Chlorate. Scheringa (*Pharm. Weekblad.*, XLVIII., 15). SO_2 reacts readily with $KClO_3$ but not with $KClO_4$. 1 gm. of the sample dissolved in 130 c.c. of water is placed in an Erlenmeyer flask and a stream of SO_2 passed over the surface. The $KClO_3$ is thus reduced to KCl, and may be estimated by $AgNO_3$ after boiling out excess of SO_2. In the filtrate separate excess of Ag by Na_2CO_3 and evaporate and ignite the residue to convert the $KClO_4$ to KCl.

Iodide in Small Quantities. Péron (*J. Pharm. Chim.*, III., 242). By adding NaOH followed by $K_2Mn_2O_8$ and boiling, the iodide is converted to $NaIO_3$. The excess of $K_2Mn_2O_8$ is destroyed by alcohol, and after cooling KI and acetic are added, and the I liberated is titrated with $N/10$ or $N/100 Na_2S_2O_3$. Chlorates or bromates should be previously destroyed by $NaHSO_3$, cyanides by boiling with acetic, since under the conditions produced they would form nitrites. NH_4 salts also should be removed before testing, by boiling with the NaOH.

Iodine and Thiosulphate. Batey (*Analyst*, XXXVI., 132). With solutions containing excess of $NaHCO_3$, titrating directly with $N/10$ I gave one result, but adding an excess of $N/10$ I and titrating back with $N/10 Na_2S_2O_3$ gave another in which more I was absorbed. The amount of I taken up appeared to be proportional to the excess of $NaHCO_3$ used. Saturating the solution with CO_2 removed the difficulty somewhat, but not completely. When borax was used instead of $NaHCO_3$ the absorption of I was much less, but some discrepancy appeared. The substitution of Rochelle salt for the $NaHCO_3$ prevented it entirely. Investigation showed that there was a temporary formation of NaOI, which oxidized the S up to Na_2SO_4.

Sulfur by Carius' Method. Annelli. With either pure S or bitumens, heating in sealed tubes with fuming HNO_3, and subsequently precipitating as $BaSO_4$, gave high results (101 to 105 per cent.). But when $Ba(NO_3)_2$ was put into the tube together with the other materials, correct results were obtained.

Sulfur in Illuminating Gas. Dickert (*J. f. Gasbeleuct.*, LIV., 182). When "perhydrol" (a concentrated solution of H_2O_2) is added to strong NaOH solution, a crystalline precipitate of $Na_2O_2 \cdot 8H_2O$ forms, which will oxidize to sulfate, H_2S and most other S compounds to be found in gas. Thiophene is not so oxidized, but as it exists ordinarily in very minute quantities, the fact is unimportant. Using a Drehschmidt absorption apparatus, and charging it with a solution made up of 10 c.c. perhydrol with 75 c.c. of 24 per cent. NaOH (sp. gr. 1.263), the gas can be passed through it in amounts required. The action is so rapid that the rate of passage of the gas may even be made 100 liters per hour, without loss. Acidifying with HCl, and precipitation by $BaCl_2$, or some other satisfactory method may then be applied to the solution.

Nitrous and Nitric Acids—Diphenylamine Test. Withers and Ray (*J. Am. Chem. Soc.*, XXXIII., 708). Numerous modes of preparing the reagent and making the test are on record, many of them being more or less indefinite as to details. The authors advise to prepare

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the reagent thus: Dissolve 0.7 gm. diphenylamine in 60 c.c. concentrated H_2SO_4 with 28.8 c.c. H_2O. Cool completely and add 11.3 c.c. concentrated HCl slowly. Allow to stand over night, when a little of the base separates out. To apply the test, take 1 c.c. of the solution to be tested, add one drop of the reagent, and mix by shaking, then run in 2 c.c. concentrated H_2SO_4 down the side of the tube so that it forms a layer below. Place in a bath at 40° for 15 to 20 minutes, when the blue color develops. By longer heating (1 hour) one part of nitrite nitrogen in 32 million, or one part of nitrate nitrogen in 44 million are detected.

Nitrates in Waters—Phenolsulfonic Method. Chamot, Pratt and Redfield (*J. Am. Chem. Soc.*, XXXIII., 366). A critical study of the method has been made. As to the reagent, it should consist only of phenoldisulfonic acid. Most methods of preparing it afford more or less phenolmonosulfonic acid, which gives greenish shades and other objectionable features. When freshly made the monosulfonic acid is not so large as results from standing, so that the age of the reagent may cause troublesome discrepancies. Gill's method of preparation (*J. Am. Chem. Soc.*, XVI., 122) was found to be the best. The authors describe later a method which gives no mono-acid. (See below.)

The amount of the reagent used makes a difference. The variations are the greater, the more of the mono-acid in the reagent. 2 c.c. was found to be the best amount.

The temperature of the water residue at the moment of applying the reagent is important. Cooling to room temperature is best. Here again the presence of mono-acid makes a difference.

The kind of alkali used has some effect. KOH was found to be the best, because giving the strongest color.

Organic matters which mask or modify the colors should be removed by use of alumina cream.

Mechanical losses of nitrates by evaporation were found to be unimportant in extent.

Carbonates in the residue cause some paling. The influence of CO_2 is most marked when the reagent contains the mono-acid. When alkaline carbonates are present, acid should be added in proportion sufficient to nearly neutralize before evaporating.

Chlorides have a decided influence. They are best removed by adding enough Ag_2SO_4 to combine with all but 0.5 part per million of the Cl, adding Al_2O_3 cream, and filtering off before evaporating.

Fe and Mg salts in some cases are present in such quantities as to interfere by preventing complete contact of the reagent with the nitrates present.

Nitrites up to 0.2 N per million have no appreciable effect, and up to 1 per million cause a practically negligible error.

The phenoldisulfonic acid containing no mono-acid can be made by heating together 25 gm. phenol, 150 c.c. concentrated H_2SO_4 and 75 c.c. fuming H_2SO_4 (13 per cent. SO_3) for 2 hours at about 100°. To obtain tripotassium nitrophenoldisulfonate for color comparisons add pure KNO_3 little by little in the cold to the above reagent in the pro-

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portion of 0.1076 gm. KNO_3 to every 1 c.c. of the acid, stirring in after each addition. Dilute, stir in $BaCO_3$ until a deep yellow color results, then filter, and extract the Ba salt with boiling water. To the solution add K_2CO_3 to convert to the K compound, filter from $BaCO_3$, evaporate and crystallize out. To prepare standards dissolve the recrystallized salt in water, and standardize by comparison with the results obtained from a known weight of KNO_3 treated in the usual manner.

Nitrates—Phenol Sulfonic Method. Caron and Roquet (*Bull. Soc. Chim.*, VII., 1021 and 1025). The absorption of moisture by the water residue after evaporation is regarded as a probable source of discrepancies in the results. Variations in the mode of preparing the reagent also produce variable results. The recommendation is to prepare the reagent with 1 c.c. liquefied phenol, and 10 c.c. concentrated H_2SO_4. Other phenolic substances can be used in the reagent, *e. g.*, a solution of 1 to 5 per cent. salicylic acid in concentrated H_2SO_4. This reagent does not keep well. It is suggested to add to 10 c.c. the water 1 c.c. of a 1 per cent. solution of sodium salicylate, and after evaporation to add 1 c.c. concentrated H_2SO_4, mix thoroughly, then dilute and proceed in the usual manner.

Nitrates—Phenol Sulfonic Method. Caron and Roquet (*Bull. Soc. Chim.*, VII., 1021 and 1025). By using a freshly prepared phenol sulfonic reagent, Cl, up to 10 parts in 100,000, is prevented from interfering. The phenol in the reagent may be advantageously replaced by salicylic acid which gives more intense colorations. The reagent had best be made up fresh. A convenient method is to evaporate 10 c.c. of the water with 1 c.c. of 1 per cent. sodium salicylate and then treat the residue with 1 c.c. concentrated H_2SO_4. After some minutes add 10 c.c. of water, and then excess of NH_4OH, or of NaOH.

Kjeldahl Method. Edwards and Chads (*Ch. News*, CIII., 138). To prevent "bumping" when using this method, granulated Zn is frequently used. This, however, often causes a troublesome "frothing." The authors find that the use of granulated Sn will also stop bumping, without also causing "frothing."

Hydrogen—Detection. Zenghelis (*Zts. anal. Chem.*, XLIX., 729). Pass the gas to be tested into a tube containing Pd or Pt sponge warmed to 80 to 120° C. H_2 is absorbed and will reduce a faintly acid solution of Na_2MoO_4, affording the strong blue color, especially if warm. Make the reagent by dissolving 1 gm. MoO_3 in NaOH, render just acid with HCl and dilute to 200 c.c. The same reaction may be produced by H_2S, SO_2, AsH_3 and PH_3 or CO. If any of these are present the gas must first be passed through a solution to absorb them as NaOH, $AgNO_3$, or $CuCl_2$ according to requirements. CH_4, C_2H_2, etc., do not interfere.

Hydrogen Peroxide—Acidity. Wöhler and Frey (*Zts. angew. Chem.*, XXIII., 2353). Endemann's conclusions are adversely criticized. The authors declare that a direct titration with phenophthalein indicator gives results very near the truth with 3 per cent. H_2O_2 solutions, rather a trifle high than too low, on account of dissolved CO_2. On 25 c.c. of

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3 per cent solutions, if 0.3 c.c. is deducted from the amount of $N/10$ alkali used, the result is correct within $\pm$ 0.1 c.c.

If the peroxide is 6 per cent. or more the H_2O_2 must be destroyed by boiling with NaOH in platinum before correct results can be obtained.

Commercial Calcium Cyanide—"Lime Nitrogen." Caro and Schück (*Zts. angew. Chem.*, XXIII., 2407). The soluble portion contains cyanamide, dicyandiamide, and urea. Treat 7 gm. of the sample with 400 c.c. of water at 20° for 2 or 3 hours, shaking frequently, make up to 500 c.c. and filter. In 250 c.c. of the filtrate precipitate cyanamide with ammoniacal Ag acetate (10 per cent. $AgC_2H_3O_2$, 4 per cent. NH_4OH). Make up to 400 c.c. and filter, reserving 300 c.c. of the filtrate for determination of the other nitrogen compounds. Wash the precipitate until free from NH_3 and in it determine N of cyanide by Kjeldahl method. To the 300 c.c. of filtrate, add KOH in excess, and boil out all NH_3. Dilute to 400 c.c., filter off the precipitate containing dicyandiamide, and use 300 c.c. of the filtrate for urea determination. The precipitate is washed, and Kjeldahled to get dicyandiamide N. In the solution containing urea the Ag is separated by H_2S, and the liquid is evaporated down and also Kjeldahled.

The N is usually distributed thus: in cyanamide 21 per cent.; in dicyandiamide 0.1 per cent.; in urea 0.7 to 0.8; in insoluble 0.6 to 0.8.

Collecting Samples of Subsurface Water. Winkler (*Zts. angew. Chem.*, XXIV., 341). The bottle to receive the sample is filled with strong brine, and is swung in a metal clip in such a manner that by pulling a tripping cord it can be inverted when lowered to the desired depth. After remaining inverted for say 10 minutes, slacking the tripping cord allows it to assume the upright position, when it can be drawn up.

Color Reactions of Minerals. Gaubert (*Bull. Soc. Franc. Min.*, XXXIII., 324). Reference is made to work in this direction by Levy, in 1887, which involved a decomposition of the mineral. The author finds that it suffices to digest the finely pulverized mineral for some hours in a test tube with concentrated H_2SO_4 in which the reagent is dissolved. Papaverine dissolved in concentrated H_2SO_4 gives a red tint, which is turned blue by V, W, Cr, or Ti oxides (acids), or brown with Sn, affording a means of distinguishing between cassiterite, rutile, and zircon. Wulfenite (Mo) gives the reaction quickly, vanadanite and crocoite (V) gives it slowly. With morphine in the reagent the Ti minerals give a crimson color. By the rapidity with which the color develops, the different Ti minerals may be distinguished.

Free Carbon in Tar. Ceruti (*L'Ind. Chimica*, XI., 5). Put 10 gm. of the tar into a 200 c.c. beaker, and heat in an air bath at 110° for two hours; then add 150 c.c. of castor oil, previously heated to 120°, and heat two hours more. Then pour the oil little by little upon a weighed filter mounted inside the bath, the temperature of 110° being maintained. Wash with hot castor oil, withdraw from the bath and cool, and then wash with alcohol and CCl_4, dry and weigh. The

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residue contains about 95 per cent. C, the most of which is of the character of lamp black, but about two per cent. of it is graphitic in character.

Benzene in Gasoline (for Automobiles). Halphen (*Mat. Grasses,* III., 1897). Benzene is used as an adulterant or partial substitute. The sp. gr. is something of a guide as indicating its presence, that of the gasoline being 0.702 to 0.730, while the mixtures show 0.750 to 0.760 and that of benzene is 0.880. For the quantitative determination the author proposes nitrating by slow addition of 10 c.c. of fuming HNO_3 (4 or 5 drops at a time) which is kept cool, the mixture being frequently shaken. After nitration, 10 c.c. of water are run in, a little at a time, and then 40 c.c. at once, when by shaking the nitrous vapors are absorbed. The oil is eventually treated in a graduated cylinder with concentrated H_2SO_4 and the increase in bulk of the acid is read off. This increase multiplied by 0.87 gives the volume of benzene present.

As most oils are acted on to some extent by H_2SO_4 alone, it is necessary to precede the operation by shaking with some concentrated H_2SO_4 and using the purified material for the test.

Mineral Oils. Park and Worthing (*Chem. Eng.,* XI., No. 4). By determining the amount of Br absorption, some information as to the character of the oil may be gained.

0.25 to 0.5 gm. of the oil (free from moisture) is dissolved in 25 to 30 c.c. CCl_4 (also perfectly dry) in a stoppered Erlenmeyer flask. 25 c.c. of a $N/10$ solution of Br in CCl_4 is then added, and the flask set aside in the dark for 30 minutes. (Drying of the oil and of the CCl_4 is effected by previous treatment with some solid $CaCl_2$.)

After the above, 100 c.c. of water and 2 gm. KI are added to the flask, and the equivalent of the unabsorbed Br titrated with $N/10 Na_2S_2O_3$ and starch indicator. This gives total bromination; then 0.5 gm. $NaIO_3$ is added, and titration with the $Na_2S_2O_3$ continued to a second disappearance of free I. The Br indicated by this last titration is the Br substitution number, because the measure of the HBr formed. The type equations would be:

Addition: $C_2H_4 + Br_2 = C_2H_2Br_2$.

Substitution: $C_6H_5CH_3 + Br_2 = C_6H_5CH_2Br + HBr$.

By subtracting *twice* the Br substitution number from that for total bromination one obtains the Br addition number.

Large addition numbers indicate acetylenes, olefines or aromatic compounds with side chains. Large substitution numbers indicate benzenes. Small total bromination indicates paraffines or naphthenes.

From such results it would be impossible to state what classes of compounds are present, but the method will serve to distinguish the type of oil in hand.

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BOOK REVIEWS.

Calcites of New York. Memoir 13 of New York State Museum. By HERBERT P. WITLOCK. Quarto. 189 pages and 27 plates.

This memoir contains a minutely detailed description of the various types of crystals of calcite from twenty different occurrences in New York State with carefully drawn plates and elaborate tabulations of forms.

An interesting attempt is made to show a connection between genetic conditions and crystal habit and certain zones of planes are shown to occur in certain localities. The proof that some one observed condition at these localities is the direct cause of the crystal habit needs strengthening by synthetic experiments and a larger series of occurrences.

The most notable coincidences pointed out are: That a zone of second order pyramids with (8.8.16.3) usually dominant occurs in the New York localities of Union Springs and Lyon Mt. and also Bad Lands in Dakota, the Belgian locality of Rhisnes and at Andreasberg, Hartz, and that at each locality the formations and associates indicate the presence of considerable silica in the crystallizing solutions producing the calcite.

Another coincidence is that in the four described New York localities in which the calcites are associated with iron ores the calcite shows a zone of negative scalenohedra.

A. J. M.

The Optical Properties of Crystals, with general introduction to physical properties. A translation of selected parts of the fourth edition of Groth's "Physikalische Krystallographie." By B. H. JACKSON, M.E., M.A. 8vo. xiv + 309 pages. Cloth, $3.50 net (15/—net). With 121 figures in the text and two colored plates. New York, John Wiley & Sons.

The portions of Groth's great work selected for translation are practically limited to the discussion of the optical properties of crystals and to the influence of other properties on the optical properties. Professor Groth assumes only that the user possesses an "acquaintance with general experimental physics and chemistry but with no mathematics beyond what is afforded by the secondary schools." "The treatment of crystal optics is based on its purely geometrical aspect, suggested by the author—Fletcher. This method makes it possible, yet without mathematical theory, to gain a correct insight into even the most complicated phenomena (conical refraction, for example), something which is indispensable for microscopical studies." It is perhaps to be regretted that Mr. Jackson did not include those portions of Part III. ("The Methods of Crystal Investigation") which deal with the practical determination of the optical characters but even without that the translation

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will be of considerable aid to those who can not readily use the German work.

A. J. M.

Tables for the Determination of Minerals. By EDWARD KRAUS, Ph.D., and WALTER HUNT, A.M. McGraw-Hill Book Company, 239 West 39th Street, New York. 254 pages, 0×9. $2.00 net.

Tables for the determination of minerals by physical tests practically all use as the first four or classifying tests, luster, color, streak and hardness, and vary chiefly in the order that these are used. The subordinate tests or characters are again practically always the same.

Between one set of tables and another the differences are not great and are partly in the order and partly in the skill with which they are printed to catch the eye.

The Kraus-Hunt tables are evidently carefully prepared and the facts conveniently and compactly marshalled. Compared with, for instance, the Frager & Brown (Weisbach) tables the notable *differences* are:

Limitations to 250 important species.

Omission of blowpipe confirmations.

More details as to associates and occurrence.

Uniform classification throughout in order—luster, color, streak, hardness—instead of metallic luster by color and hardness.

The authors have apparently enlarged the definition of hardness so that the production of a groove or scratch on a loosely compacted or granular mineral such as specular hematite, ochreous limonite, the tripolite variety of opal by, for instance, calcite proves the material to have a hardness below 3. It is perhaps a convenient enlargement and yet the fine grained hematite or tripolite will in turn polish or scratch substances of accepted hardness 4 or 5. It would seem advisable to use such a term as apparent hardness for these and others in which the ideal condition of point applied to smooth surface can not be obtained and in which the results are far below the records.

Each species name is followed by the page number of the description in the mineralogies of Kraus, Dana and Moses and Parsons.

A. J. M.

Qualitative Chemical Analysis. A Laboratory Guide. By WILFRED WELDAY SCOTT, A.M., chief chemist, Baldwin Locomotive Works, formerly professor of chemistry, Morningside College. New York, D. Van Nostrand Company, 1910. 165 pp. Illustrated. $1.50.

The initial statement in the preface of this book is that its purpose is "to furnish a practical modern guide for students studying qualitative analysis." If aided by a teacher who is in sympathy with the mode of presenting the subject, and who can correct or explain the hasty and abbreviated ways of conveying the information, a student might by its use perhaps obtain a fair knowledge of the subject. In other words, the book appears to contain the essence of the plan of instruction pursued by the author in carrying classes through qualitative analysis, the

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subject being fitted into a more extended scheme of instruction in chemistry.

A student who should attempt by himself to learn qualitative analysis, with this book as a guide, would soon find his ideas on the subject in a state of hopeless chaos, unless he could refer to some kind friend to explain and elucidate. For instance, the first part is theoretical, dealing with ionization and the theory of solution. In some places the idea seems to be that readiness of ionization constitutes the only difference between strong and weak acids, in other places the view seems to be that some ions are inherently strong as compared with certain others which are inherently weak. The whole section is worked up to an exposition of the law of mass action, the explantion of which is completely unintelligible to the average student.

The mode of applying blowpipe and film tests is well described but the method of paragraphing in that part of the book has apparently been hastily done, and is consequently confusing. More hasty work is shown in the grouping of the bases (p. 18). Au and Pt are omitted altogether, Fe appears to be indicated as one of the class precipitated as hydrates by ammonium sulfide, and the relation of Mg to the alkalies is not indicated with clearness. Similarly on page 21 ferrous hydrate is said to be green, on page 52 it is stated to be white, oxidizing to red. Without the explanations of a teacher, such discrepancies would be confusing.

A rather amusing example of hastiness occurs in the description of the special reactions of zinc. A bracketed cross reference says "(See Galvanized Iron, Smith, Newth or Remsen)," p. 51.

The condensed information of Part V., entitled "Tables of Reactions of the Bases and Useful Data," is the most valuable and helpful portion of the book. In the tables of approximate normality of reagents for use (pp. 156–7) a saturated solution of $BaCO_3$ is mentioned (normality not given). A suggestion that the reagent is used in solid form diffused in water would be more helpful.

The value of the plan pursued in many schools of chemistry—drawing a sharp line between "general" (or "elementary") "chemistry," "qualitative analysis" and "quantitative analysis," is, after all, open to serious question. The fact that the structure of modern chemistry rests largely upon definite combination by weight makes it seem more rational to introduce the balance from the start, and to intermingle the three branches much more closely than is usually the practice. Also, prominence should be given to those properties of the ions or compounds which are of service, either in quantitative, in manufacturing chemistry or in industrial applications. The plan of instruction in the Massachusetts Institute of Technology has long been on the lines suggested and has proved very successful. There is good reason to believe that it might be advantageously elaborated and extended.

E. W.

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A Course of Qualitative Analysis of Inorganic Substances, with explanatory notes. By OLIN FREEMAN TOWER, Ph.D., Hurlbut professor of chemistry in Adelbert College of Western Reserve University. Second edition, revised. Philadelphia, P. Blakiston's Son & Co., 1012 Walnut St., 1911. 8vo. 84 pp. Price $1.00.

In considering books on qualitative analysis, the question arises as to the aim and scope of the subject. Of late it appears that many teachers regard it as sufficient to teach their students but *one* method for separating and identifying each one of the more commonly occurring ions. Another and broader view is that qualitative analysis is a part of the whole plan of instruction in the chemical relations of the elements, and that it should serve as preparatory for quantitative analysis and for the study of the problems of manufacturing chemistry. This would imply a much more extended instruction in the family resemblances and individual characteristics of the different groups of elements, and in a certain measure, round out by experimental demonstration the view suggested in concrete form by the periodic table of the elements.

The point may be made that such a view calls for "qualitative chemistry" as distinguished from "qualitative analysis" and that the field is too broad to be possibly covered during the time usually alloted. We concede that the field is so broad that a chemist probably never ceases to learn new facts regarding the properties of the various elements, but somewhere in his course the student should learn the salient and more important points of qualitative chemistry, and that "somewhere" is assuredly in the qualitative laboratory.

The high pressure of our busy life in other relations has had its effect upon our schemes of instruction and the inclination of our instructors in chemistry is to cut out everything conceived to be not strictly necessary. The qualitative analysis manuals of say forty years ago gave in brief supplementary form such tests as were known for the rarer elements recognized at that time, and indicated their relationships and analogies to those better known, and (though somewhat incompletely) the groupings and what may be called the family relations of the commoner elements were indicated. There was also added a few of the tests for at least three or four of the most common organic acids, and in some cases for some of the alkaloids.

Long since, all the tests for organic acids or bases were cut out of books on mineral qualitative analysis, although acetic and tartaric acids are constantly used as reagents in such work, and in many books of the present day (as in the book under consideration) the reactions and properties of the "rarer" elements have been cut out, and also all but a very limited number of the properties of the commoner elements have been cut out. Is not it time to call a halt in the cutting out business?

With more and more of the so-called rare elements being brought into use at the present day and new applications for the compounds of the commoner elements, the student's attention should be more and more directed to their properties, instead of less and less, or he will find himself very imperfectly equipped and he will justly blame his instructors for their neglect.

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The book under consideration is a type of the form of books of instruction in qualitative analysis to which allusion has been made. In the language of the preface it "is a concise treatise along modern lines." The conciseness has been carried so far that in some places references are made to other books of instruction, which indicates that the course here given is not meant to stand by itself, but to fit into a special plan of instruction. In itself it is not a complete and adequate treatise on qualitative analysis.

Aside from the serious omissions indicated it has some good features. The ion theory is tolerably well presented. Within the limits which the author has set, or has been obliged to set for himself, much useful information and plans of procedure have been condensed into comparatively few pages, though in some few respects the directions and comments are not quite satisfactory, *e. g.*, p. 46, the method for precipitating out the calcium group. In the hands of most qualitative students this would be wasteful of reagent and imperfect in results. On p. 74 it is stated that "Cr, Ba, Sr and Ca are not present in alloys." The author ignores the use of Cr in the iron industry and in certain electrical resistance wires of the German silver type. Some efforts have also been made looking to the use of Ca in alloys. One or two essential points are omitted which always need to be hammered into qualitative students, such as the meaning of the term "excess" of a reagent and what might almost be called the corollary of the same caution, the interference caused by accumulation of considerable amounts of salts in a solution. Most students are likely to add about 50 c.c. of an acid, for instance, when the directions call for 5 c.c., and when they have, at a subsequent stage, to neutralize with some alkali, they regard the anion as entirely eliminated from their solutions, as soon as the reaction ceases to be acid. The results are often such as to prevent reasonably correct conclusions. To sum up we would say that the book appears to be adapted to a special purpose and to special conditions. In some particular cases it may be useful, but the subject is handled on lines too narrow for general adoption.

E. W.

Elements of Geology. By ELIOT BLACKWELDER, associate professor of geology in the University of Wisconsin, and HARLAN H. BARROWS, associate professor of general geology and geography in the University of Chicago. Published by the American Book Company.

This is an elementary text book intended for class use. It is prepared by competent teachers and geologists of experience who have a definite aim. Using their own words,—they have "sought to give the student (1) an understanding of the general principles and processes of the science, (2) a few of its fundamentad facts, (3) an interest in the subject, and especially (4) training in clear thinking." As a means to this end more than usual attention is given to surface processes and resultant effects. There are diagrams, maps and photographs which add greatly to the usefulness of the book as a text for beginners. The block form of sketch showing underground structure and surface

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features together are especially effective and the topographic maps in standard colors are equal to the illustrations of more pretentious publications.

One readily sees that there are some innovations. For example, very little attention is given to earthquakes and volcanoes—not more than two or three pages altogether—whereas most text books devote at least a chapter to each. Likewise structural phenomena such as folds and faults are given much briefer treatment than usual. On the other hand the work of water in developing topography and the work of ice are greatly expanded—forty pages being given to glaciers. About two fifths of the 475 pages of the volume is given to historical geology.

The authors have intentionally reapportioned the topics of geology in this way in the belief that the material to which they have given most emphasis is best calculated to reach the aim set for themselves in training students to think clearly on geologic questions. Independent teachers may differ materially on this point and especially in regions where the geologic surroundings themselves emphasize other matters. For example, glaciers and glacial drift might not appear so important to one unable to reach the drift with his classes. It would seem that igneous and metamorphic processes had not been given enough credit for the very great influence they have had on the development of the earth's crust as we know it.

The book is accurate, well written, beautifully illustrated, and well suited to the use of beginners in geology.

CHARLES P. BERKEY.

Power. By CHARLES E. LUCKE, Ph.D., professor of mechanical engineering, Columbia University. New York, The Columbia University Press, 1911. 316 pages, 223 illustrations.

The material contained in this book was comprised in the Hewitt course of eight lectures, delivered by the author at Cooper Union, in 1909–1910. The subjects treated are: The relation of mechanical power and machinery to social conditions; means employed for the substitution of power for the labor of men; essential elements of steam-power systems; principles of efficiency in steam-power systems; processes and mechanism of the gas-power system; adaptation of fuel for the use of internal combustion engines; water-power systems and basal hydraulic processes; social and economic consequences of the substitution of power for hand labor.

While Dr. Lucke's work is not intended to be a *treatise* on mechanical engineering or on power plant, it will certainly be read by engineers with both interest and profit. The subject matter is presented in a form readily intelligible to the non-technical reader or student. It will serve admirably as an introduction to more detailed courses in mechanical engineering, and, since it covers a range of industrial interests respecting which everyone should have at least a general knowledge, it will be found useful also as a text-book for academic college students. The list of subjects given above sufficiently indicates the wide range of the book. After laying down the principles of the

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application of steam to power purposes, in Chapters II., III. and IV., the author next takes up the development of gas-power machinery, together with a clear and interesting discussion of the modes of using the appropriate solid and liquid fuels, for this comparatively new and important branch of power generation. The water power systems are well outlined in Chapter VII.

In presenting the industrial summary of the United States, contained in Tables III. and IV. (p. 13), it may be that the author intended to include only raw materials of manufactures and their products; but coal is certainly as much a "raw material" as farm produce, both in their way being converted into "power." In 1909, our coal output was 444,203,800 tons, worth at the mines about $580,444,300. Adding this amount to that given for mine products (Table IV.), the total reaches $1,051,562,481, which would materially alter the relative percentages of the different industries. In 1909, the value of the *metal* product alone was in this country $788,456,622.

The proof-reading has evidently been carefully done, the typography is good and there is an excellent index. While many of the illustrations are on a rather small scale, they are well chosen and ample in number and scope.

Particularly in the first and last chapters, but frequently also throughout other parts of the book, the author discusses the development of human industry in a broad, philosophical style, that makes decidedly good reading. Chapter VIII., on the social and economic consequences of the application of "power" for performing the world's work, is specially to be commended to the general reader, as well as to the engineering student.

R. P.

Die binären Metallegierungen. By K. BORNEMANN, Privatdozent, Technische Hochschule, Aachen. Part 1. Halle a. S., Wilhelm Knapp.

Dr. Bornemann has collected and brought up to date our knowledge of binary alloys, more especially their thermal diagrams, and published it in the recent numbers of *Metallurgie*. The present volume is reprinted from that paper and embraces the alloys of the alkali metals, the alloys of copper, of silver and of gold. The alloys of the other metals will follow in parts 2 and 3.

The appearance of this book is very welcome and will be especially so to those who can not readily gain access to those foreign publications in which metallographic research is published. Even for those who have extensive library facilities this collection will prove a great saving in time, and gives in a very handy form a vast amount of very valuable information.

W. C.

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International Journal of Metallography. Central periodical for the entire theory and practice of alloys. Edited by DR. W. GUERTLER. Berlin, Gebrüder Borntraeger. Part 1. 80 pp. Illustrated. 20 marks.

The first number of Vol. 1 appeared in February and contained three papers: The Aim and Scope of Metallography; Über Spannungen in Kaltgerechten Metallen, by Heyn and Bauer; Sodium Silver Alloys, by C. H. Mathewson. These are followed by sixteen pages of bibliography for the year 1910.

The study of metallography has reached such importance that the time is ripe for a publication devoted solely to it. It is often said that metallography is too theoretical. But as the editor says "the future of metallography will appear so much the more propitious in proportion as the theoretician ceases to shun the special questions associated with practice, and the technician seeks to place his empirical experiences upon a rational basis by becoming acquainted with fundamental theoretical principles."

The field covered by metallography is a very wide one, as a glance at the bibliography will show, and if this new journal carries out its plans, it will undoubtedly have all the success it unquestionably deserves.

W. C.

Mitteilungen aus dem Institut für Metallographie und Probierkunde an der Konigl. Sachsischen Bergakademie Freiberg. By K. FRIEDRICH, Professor of Metallurgy, Konigl. Technische Hochschule Breslau, 1910. Halle a.S.: Wilhelm Knapp. Part 1, pp. 138, XXIV pl.

The work here collected was done between 1905 and 1909, in the Metallographic and Research laboratories of the Bergakademie at Freiberg, and published in *Metallurgie.* Fifteen titles include the following: Copper and Arsenic, Lead and Sulphur (with A. Leroux), Notes on Liquation: Lead and Arsenic; Silver and Arsenic (with Leroux), On a Gas Crucible Furnace for Metallographic Research, Silver and Silver Sulphide (with Leroux), Lead and Silver, Zinc and Arsenic (with Leroux), On the Quantitative Determination of Small Amounts of Silver and Gold, On the Gold Contents of Samples and Springs of German Salt Deposits, On the Condensation of Furnace-fume, Iron and Arsenic, Nickel and Arsenic, Copper, Silver and Lead (with Leroux).

Professor Friedrich has shown us in particular that the constitution of mattes and speisses follows the same laws of that of alloys, that they are all solutions. His work is of interest not only to the metallographer working in the research laboratory, but also to the furnace-man out in practice. The latter will specially welcome this volume because we have, in handy form, work which is scattered in the original through several volumes. Professor Friedrich is to be congratulated in his contribution to the so-called new metallurgy.

W. C.

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A Concise History of Chemistry. By T. P. HILDITCH. New York: D. Van Nostrand Company, 1911. Cloth, 12mo. ix + 263 pages. $1.25 net.

The contents of this book are: The Evolution of the Science; The Chemical History of Fire, Air, and Water; The Ultimate Constitution of Matter; Inorganic Compounds and the Laws of Chemical Combination; History of the Elements and their Chief Compounds; History of Organic Chemistry; Compounds and Reactions in Organic Chemistry; Chemistry of Plant and Animal Life; Application of Chemistry to Manufactures; History of Physical Chemistry; The Progress of Experimental Method; Biographical Index of Chemists; Chronological Summary of Chemical Events of Outstanding Interest.

Naturally no one chemist could handle all these topics with an equally firm grasp, and the present author is most at home in the theoretical chapters, which are good examples of intelligent condensation and are brought well down to date. The discussion of the chemistry of plant and animal life deals mainly with the structure of natural organic compounds and but little with the influence of chemistry upon the actual practice of agriculture and medicine, and the student might easily be misled by the statement in the chapter on manufactures that the production of sugar in the West Indies is now "a decaying industry" or by the names selected for mention in the case of some of the other industries. On the whole, however, the author has carried out very successfully the task which he sets for himself in his preface: "This book is an attempt to outline as briefly and succinctly as possible the historical development of chemistry, and is designed more especially for those students whose interest in this aspect of the science is stimulated by the inclusion of Historical Chemistry in the syllabus of examinations which concern them." Making no attempt to acquaint the student with the personalities of the great chemists and giving comparatively little space to their reasoning or the details of their experimental evidence, but emphasizing the orderly presentation of the historical sequence of facts, the author has succeeded in recording an immense amount of material in a very compact and convenient form as may be judged by the fact that the combined indexes of names and subjects require 17 double-column pages. The compact form and numerous tabular summaries make the book very convenient for quick reference. It is written in a clear and readable style and the mechanical make-up is unusually good for a book of such moderate price. H. C. S.

Iron and Steel (a pocket encyclopedia), including allied industries and sciences. By HUGH P. TIEMANN, B.S., A.M. McGraw-Hill Book Co. 4 × 7, 354 pp., ill.

This small pocket book is a dictionary and a handbook combined. It is written in a clear brief style by one who is sure of his ground and has learned his metallurgy in the mill as well as from books. It contains a vast amount of information which till now has only been obtainable by digging in the ponderous treatises and transactions devoted to

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iron and steel. In addition the terms of the mill-man and metallographer are translated into plain everyday language.

The author is to be congratulated for having given us a book which will be valuable to us whether we be a millman, a metallographer or a metallurgist, a salesman or a student—to all that great host who are interested in the greatest of all industries, the iron and steel business.

W. C.

A Year in a Coal Mine. By JOSEPH HUSBAND. Houghton Mifflin Co., The Riverside Press, Cambridge, Mass., 1911. 171 pages.

The author of this very interesting little book describes how as a young graduate he goes to work in a coal mine, in the Middle West, with the intention of learning by actual work, the "operating end" of this great industry, so that the practical knowledge obtained would be of benefit to him in his business career. His notes of the trials and vicissitudes of a tenderfoot in a coal mine are expressed in a feeling manner and his description of the general operations of a large mine are both interesting and instructive. In the latter part of the book he brings the reader face to face with conditions which exist in a coal mine at the time of a great disaster. Many who see in the papers accounts of these diseasters hardly realize what they mean, but after reading this book one cannot help feeling that he knows more about the actual life of a coal miner, the dangers to which he is continually exposed and the manner and cause of explosions.

A. L. W.

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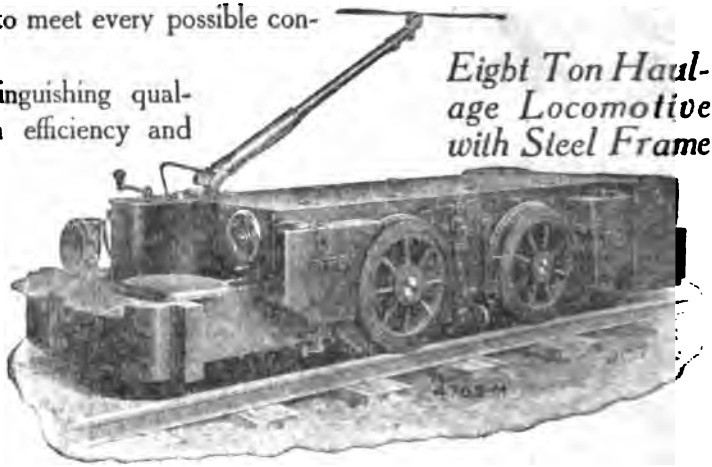

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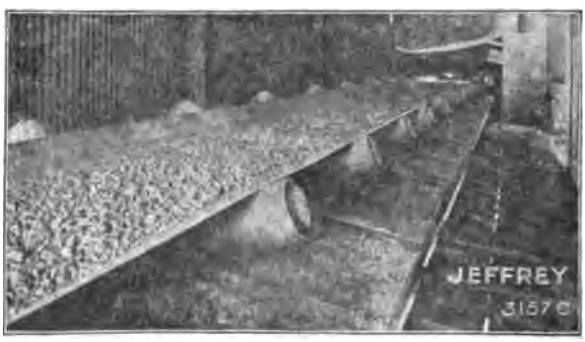

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